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# **METAL PROCESS ENGINEERING**

MIR PUBLISHERS MOSCOW



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В. ЖАДАН, Д. ВАСИЛЬЕВ**

# **ТЕХНОЛОГИЯ МЕТАЛЛОВ**

*Под общей редакцией*  
**проф. П. ПОЛУХИНА**

**СТРОЙИЗДАТ  
МОСКВА**

P. POLUKHIN, B. GRINBERG, S. KANTENIK,  
V. ZHADAN, D. VASILYEV

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# METAL PROCESS ENGINEERING

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*Under the editorship  
of Prof. P. Polukhin*

*Translated from the Russian  
by*  
**NICHOLAS WEINSTEIN**

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The course *Metal Process Engineering* is intended for engineers as a basis for the successful mastery of a number of special subjects.

An acquaintance with the methods employed to produce ferrous and nonferrous metals and their alloys, and a knowledge of their principal properties and processing procedures are of vital importance for expedient selection and proper application of metallic materials. Engineers in all lines of industry must know what effect heat treatment and other processes have on the properties of metals and what can be done to alter these properties in the desired directions.

This book has been recommended for republication by Indian specialists under the programme of the Joint Indo-Soviet Board to make the best Soviet textbooks available for Indian students.

*На английском языке*

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# THE MANUFACTURE OF FERROUS METALS, ALUMINIUM, COPPER AND TITANIUM

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## *Chapter 1*

### FUELS AND REFRACTORIES OF PROCESS METALLURGY

#### 1-1. Fuels

Metals are extracted from their ores and undergo subsequent refining processes in various types of shaft and flame furnaces in which high temperatures are developed by fuel combustion.

All fuels are of organic origin; their combustible constituents are carbon, hydrogen and various compounds of these elements (hydrocarbons). Fuels also contain a certain amount of oxygen and sulphur. Sulphur is the most harmful constituent of fuel since in the process of smelting some of it may dissolve in the metal thus impairing its properties. Fuels used in metallurgy should contain the least possible amount of inert mineral admixtures (ash) which are incombustible and reduce the calorific value.

Solid, liquid and gaseous fuels are employed in process metallurgy.

**Solid fuels.** Solid fuels include charcoal, coke and coal (both bituminous and anthracite types).

*Charcoal* is obtained by the dry distillation of wood (heating without the admission of air) in compartment kilns or stalls. It contains 80 to 90 per cent solid (nonvolatile) carbon, 10 to 18 per cent volatile matter (including moisture), 0.6 to 1.2 per cent ash and a negligible amount of sulphur. The calorific value of charcoal ranges from 6500 to 7500 kcal per kg. Charcoal is highly porous (up to 70 per cent); it burns well but has a low compressive strength (20 to 40 kg per sq cm). Charcoal is used in small-volume blast furnaces for making high grades of cast iron with a low sulphur content.

Charcoal is an expensive fuel; its production is limited because of the shortage of timber.

**Coke** is the solid carbonised residue formed in the process of heating (dry distillation) of certain grades of coal in retort (coke) ovens at  $1000^{\circ}$ - $1100^{\circ}\text{C}$ . As a rule, coke contains from 82 to 88 per cent solid (nonvolatile) carbon, 10 to 15 per cent ash and 0.5 to 1.8 per cent sulphur. The chemical composition of coke is determined by the nature of the coal from which it was obtained.

Coke possesses ample porosity (up to 50 per cent) and excellent combustibility; its lumps are grey-black in colour, sometimes they are dull grey and even slightly silvery. It is considerably firmer than charcoal, having a compressive strength as high as 140 kg per sq cm.

Coke is the chief type of metallurgical fuel for blast furnaces and cupolas; very high temperatures may be developed in its combustion. Coke has a calorific value of 6500 to 7500 kcal per kg.

**Coal** is of vegetable origin; it is a product of the decomposition of wood and plant residues accumulated under a layer of earth in remote geological epochs. Coals from various deposits may differ greatly in their physical and chemical properties; their combustible portion has the following composition: 76 to 90% C, 4 to 5.5% H and the remainder O+N+S. They contain from 12 to 20 per cent ash or even more.

Of the numerous kinds of coal, the best qualities are found in *anthracite*. It has a denser structure and contains little volatile matter. The calorific power of various coals and anthracite ranges from 4500 to 8000 kcal per kg.

Coals, including anthracite, are practically unfit for smelting iron in a blast furnace because of their low gas permeability and because they melt down or splinter at high temperatures.

**Liquid fuels.** *Masout* is employed as a liquid fuel for open-hearth and heating furnaces. It is the liquid residue of crude petroleum after driving off the lighter fractions—gasoline, kerosene and gas oil—by distillation. Masout contains 80 to 87% C, 12 to 14% H, 1.2% O+N, and 0.3% ash; it has a calorific value up to 10,000 kcal per kg. Low-sulphur grades of masout (less than 0.4-0.7% S) are used in steelmaking.

Masout is a valuable raw material of the chemical industry, and therefore it should be replaced by gaseous fuels in process metallurgy.

**Gaseous fuels.** The gaseous fuels are coke-oven, producer, blast-furnace and natural gases.

**Coke-oven gas**, obtained as a by-product in the manufacture of coke has a calorific power up to 4500 kcal per cu m and contains about 57%  $\text{H}_2$ , 22%  $\text{CH}_4$ , 6 to 7% CO and very little  $\text{N}_2$  (up to 7-8%). It burns with a colourless flame and enables very high temperatures to be developed in the furnace.

*Producer gas* is obtained in a producer furnace upon incomplete combustion of solid fuels. Its composition is: 5 to 8%  $\text{CO}_2$ , up to 30%  $\text{CO}$ , 2 to 3%  $\text{CH}_4$ , 10 to 15%  $\text{H}_2$ , while the remainder is  $\text{N}_2$  and water vapour.

*Blast-furnace gas* is a by-product of the blast-furnace process in pig iron production. After being cleaned of dust it is extensively applied in metallurgical plants both in the pure form (for firing boilers, coke-oven batteries, blast-furnace stoves, etc.) and in a mixture with coke-oven gas (for open-hearth furnaces, soaking pits of rolling mills, etc.). The approximate composition of blast-furnace gas is: 12%  $\text{CO}_2$ , 28%  $\text{CO}$ , 0.5%  $\text{CH}_4$ , 2.5%  $\text{H}_2$  and 57%  $\text{N}_2$ . Its calorific value ranges from 850 to 1000 kcal per cu m.

*Natural gas* is obtained either from gas fields or from those indigenous to an oil deposit.

Natural gas is a very cheap fuel with a high calorific power and is rich in hydrocarbons, especially methane. Its approximate composition is 93%  $\text{CH}_4$ , 2%  $\text{CO}_2$ , 1%  $\text{N}_2$ , 1%  $\text{H}_2$  and 3%  $\text{C}_m\text{H}_n$ . The calorific value is about 8000 kcal per cu m. In recent years metallurgical plants have begun to use natural gas to a great extent.

## 1-2. Refractories

The internal parts of all metallurgical furnaces, stoves and ovens are lined with refractory materials by laying, ramming or burning-in operations.

Metallurgical furnace refractories must withstand high temperatures; have sufficient mechanical strength and hot resistance against the chemical attack of liquid slags, molten metals and gases; retain a constant volume and possess a definite porosity.

In metallurgical plants refractories are employed in the form of powder, bricks and special shapes—stopper heads, sleeves, etc.

As to their chemical properties, refractories are classified as acid, basic and amphoteric, or neutral, depending upon the chemical nature of their constituent oxides.

*Acid refractories* include quartzite and quartz sand which are of rock origin and contain from 93 to 100 per cent silica ( $\text{SiO}_2$ ). Ground quartzite containing from 93 to 97 per cent  $\text{SiO}_2$  is used to make dinas brick for lining the walls, bottoms and roofs of open-hearth, electric and certain other furnaces. The melting point of dinas brick (also called silica brick) is  $1690^\circ\text{--}1730^\circ\text{C}$ ; its softening temperature is  $1550^\circ\text{C}$ . The bottoms of acid open-hearth furnaces are burned in and renewed with ground quartzite and white quartz sand.

*Basic refractories* include calcined dolomite and calcined or fused magnesite.

The calcined dolomite used in metallurgy is obtained by heating raw dolomite rock  $\text{MgCO}_3 \cdot \text{CaCO}_3$  to a high temperature ( $1600^\circ\text{C}$ ). The  $\text{CO}_2$  is driven off in calcinating the raw dolomite and a refractory material is formed which contains 52 to 58%  $\text{CaO}$  and 35 to 38%  $\text{MgO}$ . Calcined dolomite can withstand heating up to  $1800^\circ\text{C}$ - $1950^\circ\text{C}$ .

Calcined magnesite is the result of the high-temperature processing of raw magnesite rock  $\text{MgCO}_3$ . Calcining proceeds according to the reaction  $\text{MgCO}_3 \rightarrow \text{MgO} + \text{CO}_2$  from which  $\text{MgO}$  is obtained. The melting point of this magnesium oxide ( $\text{MgO}$ ) is over  $2000^\circ\text{C}$ . The highest refractory properties are found in fused magnesite which is obtained from raw magnesite at the fusion temperature (over  $2000^\circ\text{C}$ ) of the calcined product.

Dolomite and magnesite brick and other products of these materials are extensively applied in lining smelting and heating furnaces. The bottoms of basic open-hearth furnaces are burned in and renewed with magnesite and dolomite granular products.

Chrome-magnesite refractory bricks possess high resistance to thermal shock. They contain about 65 to 70 per cent calcined magnesite ( $\text{MgO}$ ) and 25 to 28 per cent chromite ( $\text{FeO} \cdot \text{Cr}_2\text{O}_3$ ). This brick is used chiefly to build the hanging arches of open-hearth furnaces.

Calcined fireclay, containing about 30%  $\text{Al}_2\text{O}_3$ , belongs to the group of *neutral refractories*. The main constituent of raw fireclay is the mineral kaolinite  $\text{Al}_2\text{O}_3 \cdot 2\text{SiO}_2 \cdot 2\text{H}_2\text{O}$  from which it obtains its plastic properties. In calcining, the kaolinite loses its combined water while the alumina ( $\text{Al}_2\text{O}_3$ ) and silica ( $\text{SiO}_2$ ) go over into the calcined mass called *grog* or *chamotte*.

Grog withstands heating up to  $1610^\circ\text{C}$ - $1670^\circ\text{C}$ . A mixture with sufficient bonding strength to mould various grog refractory products such as brick, sleeves, stopper heads, etc., is made of ground grog (50 to 60 per cent) and raw fireclay (50 to 40 per cent). The moulded and dried shapes are baked at  $1300^\circ\text{C}$ - $1400^\circ\text{C}$  after which they become suitable for application in the metallurgical industries. Chamotte brick containing 55 to 60%  $\text{SiO}_2$ , 34 to 38%  $\text{Al}_2\text{O}_3$  and 1.5 to 3%  $\text{Fe}_2\text{O}_3$  is employed in both ferrous and nonferrous metallurgy. Ladles for molten metal are lined with this brick; it is used for the lining and checkerwork of blast preheating stoves, blast furnaces, etc. Grog is also used for the refractory parts of ladles employed in pouring steel and for certain other articles (e. g., crucibles).

Recently, high-alumina chamotte refractories ( $\text{Al}_2\text{O}_3 > 40$  per cent) have come into use in process metallurgy. They have a very high chemical stability and refractoriness.

## Chapter 2

### THE MANUFACTURE OF PIG IRON

The raw materials for the production of pig iron in blast furnaces are: (1) iron and manganese ores, (2) fuel and (3) flux.

#### 2-1. Iron and Manganese Ores

**Iron ores.** Iron ore comprises the ore body and the dead rock or gangue.

The ore body is the name given to the natural chemical compounds of iron, most frequently its oxides, found in the ore. Practically all of the iron (98 to 99 per cent) is reduced from its chemical compounds in the blast furnace and goes over into the pig iron. The gangue of iron ore may be of various chemical composition. Most frequently it consists of quartzite ( $\text{SiO}_2$ ) or sandstone with an admixture of clayey matter ( $\text{Al}_2\text{O}_3 \cdot 2\text{SiO}_2 \cdot 2\text{H}_2\text{O}$ ) and less often of limestone ( $\text{CaCO}_3$ ) or dolomite ( $\text{CaCO}_3 \cdot \text{MgCO}_3$ ). In the blast furnace the gangue is converted into slag thereby separating from the pig iron.

Iron ores are classified as rich or lean according to the amount of gangue they contain. After crushing and sizing, rich ores are charged directly into the furnace while lean ores are subjected to concentration in which the proportion of the ore body (iron oxides) in the ore is increased. As more effective concentration techniques are developed, it becomes possible to use lean ores which were previously dumped or not worked at all.

A certain amount of harmful impurities are always present in iron ores; they include sulphur, arsenic and phosphorus. Deposits mined first on an industrial scale are those in which the iron ore contains a negligible amount of harmful impurities and the optimum amount of gangue for the given conditions.

The metal-bearing components of the charge that are of the greatest commercial importance in pig iron production are hematite, limonite, magnetite and siderite iron ores, composite and manganese ores and metallurgical waste.

*Red iron ore, or hematite,* is an ore in which the iron is in the form of anhydrous ferric oxide  $\text{Fe}_2\text{O}_3$  (theoretical composition: 70% Fe and 30%  $\text{O}_2$ ). The iron content of this ore is very high—from 55 to 65 per cent—while that of harmful impurities, sulphur and phosphorus, is very low. Hematite may vary in colour, from dark red (cherry red) to dark grey. The gangue in hematite is most often quartzite.

The physical properties of hematite, its density and mechanical strength, may vary in a wide range; its reducibility in the blast furnace is quite good.

*Limonite*, or *brown iron ore*, contains from 35 to 55 per cent iron in the form of hydrous oxides and most frequently in the form of the mineral limonite  $2\text{Fe}_2\text{O}_3 \cdot 3\text{H}_2\text{O}$  with a theoretical composition of about 60% Fe and 14% combined (hydrate) water.

The colour of this ore may vary from yellow to brownish yellow. The hydrate water in limonite ore is driven off at high temperatures; this makes the ore porous and improves its reducibility.

Limonite is not a firm ore; in mining, handling and transporting, it forms a great deal of fines and dust (especially after drying). Certain ores of the limonite type contain a great deal of phosphorus.

*Magnetite* contains from 50 to 69 per cent iron; its ore body is ferri-ferrous oxide  $\text{Fe}_3\text{O}_4$  which displays pronounced magnetic properties (theoretical composition: 72.3% Fe and 24.6%  $\text{O}_2$ ).

The gangue of magnetite ore is of a siliceous nature with an admixture of a certain amount of other oxides.

Magnetite is the most dense of the iron ores; its colour is dark grey or black in various shades.

It is more difficult to reduce magnetite than other iron ores; in some cases it contains much sulphur (up to 1.5-2 per cent) and sometimes such objectionable impurities as zinc which destroys the furnace linings.

*Siderite*, or *spathic iron ore*, contains from 30 to 40 per cent iron in the form of ferrous carbonate  $\text{FeCO}_3$  (theoretical composition: about 48% Fe and 37%  $\text{CO}_2$ ). The gangue of siderite ore consists of  $\text{SiO}_2$ ,  $\text{Al}_2\text{O}_3$  and a small amount of  $\text{MgO}$ . Certain siderite ores contain a sandy-argillaceous gangue.

The colour of natural siderite is yellowish white or grey in various shades.

Siderite is easily oxidised in air where it loses its  $\text{CO}_2$  and is transformed into  $n\text{Fe}_2\text{O}_3 \cdot m\text{H}_2\text{O}$ .

Before being charged into a blast furnace, siderite ore is usually calcinated according to the reaction  $\text{FeCO}_3 \rightarrow \text{FeO} + \text{CO}_2$ . This removes the hygroscopic water and the carbon dioxide  $\text{CO}_2$  combined with the iron. The ore becomes very porous and brittle; it is easily crushed without forming much dust. In some cases raw, uncalcinated ore is charged into the blast furnace.

Siderite possesses the highest reducibility of all iron ores.

*Composite iron ores* contain other useful metals besides iron which pass into the pig iron in the blast-furnace process. These metals alloy the pig iron and improve many of its properties. Among the more valuable composite iron ores are chrome-nickel iron ore and chromium iron ore from various deposits. Chromium iron ore with a high

$\text{Cr}_2\text{O}_3$  content is used in the refractories industry; ores poorer in chromium are used in smelting blast-furnace ferrochromium.

**Manganese ores.** Since most iron ores contain very little manganese it is necessary to add manganese ore to the blast-furnace charge in pig iron manufacture.

The ore body of a manganese ore may comprise one or several oxides of manganese, namely:  $\text{MnO}_2$  (manganese peroxide or pyrolusite),  $\text{Mn}_2\text{O}_3$  (manganic oxide or braunite),  $\text{Mn}_3\text{O}_4$  (mangano-manganic oxide or hausmannite) and compounds of manganese oxides and oxides of other elements.

Manganese ores used in the blast furnace contain from 25 to 40% Mn.

**Metallurgical waste.** In addition to iron ore the blast furnace is charged with a certain quantity of metallurgical waste, namely: (1) furnace flue dust (containing 30 to 45% Fe and 12% C) which undergoes lumping before use, (2) scrap collected in the casting bed and near the pig casting machine of the blast-furnace department, (3) steel-making slags of high manganese content (containing from 10 to 18% Fe and 6 to 10% Mn), and (4) scale and welding slags of the rolling and forging departments.

## 2-2. Fuels

Only solid fuels are employed in pig iron manufacture. Coke is used in over 98 per cent of pig iron production while the use of charcoal is insignificant (1 to 2 per cent).

Fuel plays an important role in the blast-furnace process. Its function is twofold; first, to evolve enough heat in its combustion to attain the high temperature required for reducing the ore and for melting and superheating the pig iron and slag that are formed; and second, to supply carbon that directly participates in the ore reducing reactions.

Coke used in blast-furnace operation must have a high calorific value, ample strength, good porosity and it must contain a minimum amount of harmful (sulphur and phosphorus) and mineral (ash) impurities.

## 2-3. Fluxes

The functions of the flux are to lower the melting point of the ore gangue and to fuse it with the ash of the fuel.

The most commonly used blast-furnace flux is limestone ( $\text{CaCO}_3$ ); dolomite ( $\text{CaCO}_3 \cdot \text{MgCO}_3$ ) is employed less frequently.

In some cases siliceous alumina fluxes are charged into the blast furnace and in others, only silica.



The quantity of a definite type of flux required for operation is determined by computations based upon the chemical compositions of the gangue and ash, and upon the requirements made to the physicochemical properties of the slag formed in the blast furnace.

Blast-furnace fluxes should contain a minimum amount of the harmful impurities—sulphur and phosphorus. Silica is an undesirable constituent of limestone because it decreases the fluxing power and increases the volume of slag in the blast furnace.

## 2-4. Preliminary Treatment of the Raw Materials

Raw materials for pig iron manufacture—iron ores, coke and fluxes—undergo appropriate treatment before they are charged into the blast furnace.

*Coke* requires the simplest preparation since only the fines need be removed by means of roller screens (revolving disk grizzlies).

*The fluxing materials* are subjected to crushing in jaw or roll crushers and then the fines are removed in bar (vibrating) or drum (rotary) screens.

The preliminary treatment of *iron ores* for smelting depends upon their iron content and their physical properties. Lump ores, sufficiently rich in iron, are delivered to crushing and sizing plants where they are first classified by screens into three fractions: coarse, medium and fine. Coarse lumps (over 60 to 100 mm in size) are then crushed to the medium size and screened again.

The medium fraction (30 to 80 mm lumps) of the ore is shipped by railway or waterways and delivered to the ore yard of the metallurgical plant where a stock equal to two or three months' consumption of the blast-furnace department is stored.

Iron ores are stored and handled in the plant according to a definite system which enables their composition to be equalised by mixing. This is done by dumping the ore, after unloading it from the cars or ships, in shallow horizontal layers in high piles which are subsequently restacked into new piles. The grab crane used in restacking the piles removes the ore in vertical slices thereby mixing the ore and equalising its chemical composition.

If the iron ore has an insufficient iron content it undergoes concentration before being shipped to the metallurgical plant.

Several methods of iron ore concentration are in use. Limonite, or brown ore, with a sandy-argillaceous gangue is washed with water. A high-speed jet separates the lighter particles of sand and clay from the ore body and carries them away. The gangue of brown ore is separated from the ore body in log washers, revolving cylindrical or conical trommels or in jigging machines with a stationary or shaking

screen and a pulsating current of upward-flowing water. After natural or artificial drying the washed ore is suitable for smelting.

Lean ores containing disseminated magnetite ( $\text{Fe}_3\text{O}_4$ ) are concentrated by means of magnetic separators of the drum or belt type.

The following procedures for magnetic concentration of iron ores are accepted practice:

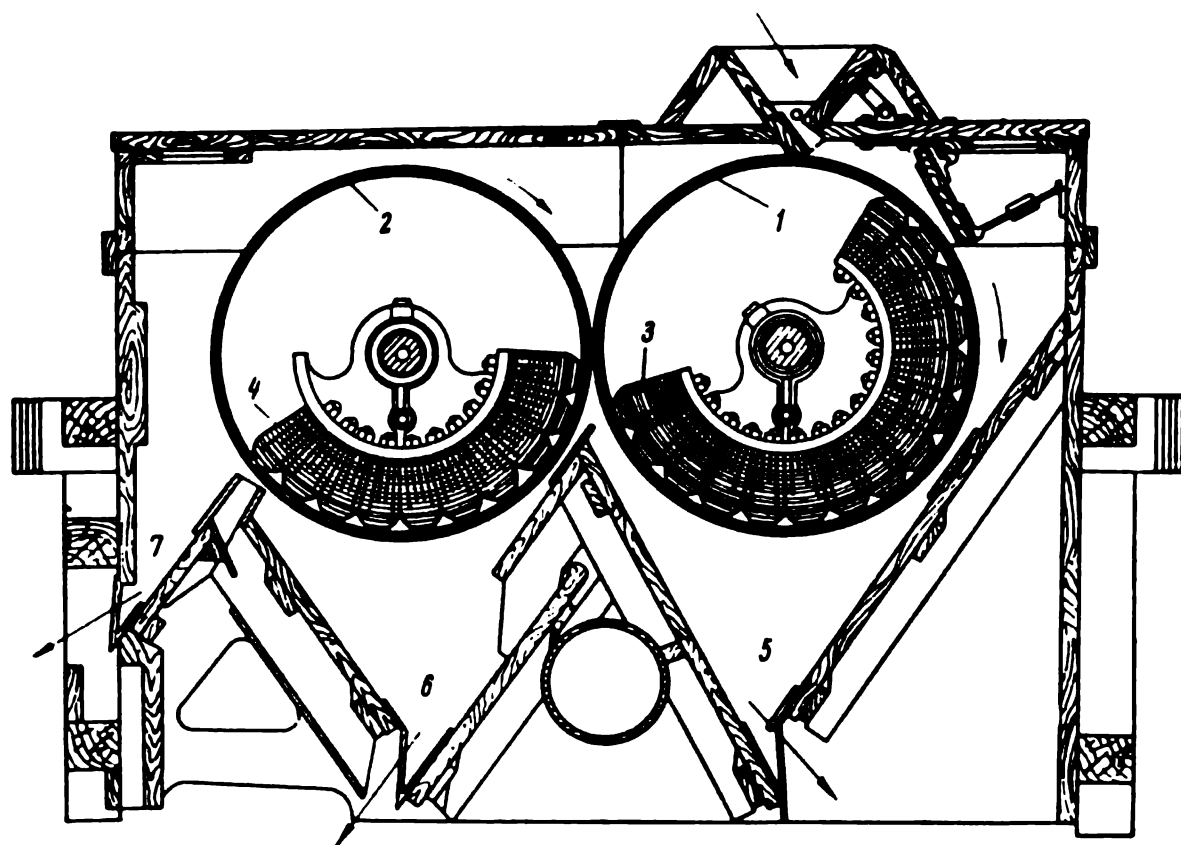


Fig. 1. Principle of the magnetic separator for ore concentration

(1) Ore with large and medium magnetite inclusions is crushed to lumps 25 to 30 mm in size and then undergoes dry magnetic separation.

(2) Ore with medium and fine inclusions is first ground into particles 3 mm in size and then undergoes wet magnetic separation.

A separator for the dry concentration of magnetite ores is illustrated in Fig. 1. It comprises two hollow drums 1 and 2 made of a non-magnetic material (brass, plastic, etc.) which rotate clockwise about stationary shafts mounting radial multiple-pole electromagnets 3 and 4. The oppositely charged poles of the electromagnets (north and south) alternate and face the inner surface of the drum. When current is passed through the electromagnets they produce a magnetic field, near this surface, which is of the corresponding intensity and polarity.

The iron ore prepared for concentration is fed through the hopper and runs down over the outer surface of the first rotating drum. The

ferromagnetic constituents of the ore are attracted to the drum surface and are carried by it in the clockwise direction. They roll over with each change in polarity and thereby free themselves from the adherent nonmagnetic particles.

The nonmagnetic fraction of the iron ore, its gangue, drops away from the first drum and runs down into hopper 5 for nonmagnetic materials which are called tailings. After passing out of the magnetic field of the first drum, the ferromagnetic particles are thrown by inertia to the surface of the second drum on which a weaker magnetic field has been set up. Consequently, lumps and grains of ore which have the highest magnetic permeability are attracted to and held by this drum until they pass out of its field and are thrown into hopper 7. The latter accommodates the finished product of dry magnetic concentration which is called the concentrate.

The weakly magnetic fraction of the ore drops away from the second drum and into hopper 6 from where it is delivered for grinding and subsequent wet magnetic separation.

Wet magnetic separation of finely ground iron ores is carried out in a separator in which the lower half of the drum is sunk in a cylindrical box and its surface is washed with a stream of water.

The bottom of the box is made of brass sheets and is arranged near the outer surface of the rotating drum leaving only a fairly narrow gap into which the ore to be concentrated and the water are fed. The magnetised ore grains are attracted to the drum surface and are carried upwards to the unloading zone where they are removed by a magnetic brush and directed to the concentrate hopper.

Fine fractions of iron ore, formed in crushing or obtained in screening lump ores; the concentrate obtained after wet concentration of magnetic ores; and extremely fine (dustlike) ores can be used in smelting only after their *conglomeration*.

A number of methods are used for conglomerating ore fines and dustlike ores. The simplest of them is *briquetting* which involves compacting fine material in a mould either after adding a bond—clay, water glass, resin or cement—or without a bond. The compacted briquettes acquire the desired strength after air drying or high-temperature baking. Ore briquettes are good material for pig iron smelting but their production is complicated and the output is low.

A promising method of preparing iron-ore dust and finely ground concentrates for smelting is *nodulising*. Nodules are obtained by mixing the dustlike ore mass with a very small quantity of the appropriate bonding material—finely ground clay, lime, etc.—and wetting the mixture to an 8 to 10 per cent moisture content. This mixture is fed into the mixer which may be either a rotary shallow inclined bowl or a hollow drum. Upon rotation of the inclined bowl or the drum, the moistened ore charge rolls over and over inside the drum or bowl,

first forming small lumps which then grow into balls (nodules) 25 to 30 mm in size.

The size of ore nodules formed in the mixer can be controlled within certain limits by regulating the thickness of the layer of the ore charge, moisture content, amount of bonding material and the rotary speed of the drum or bowl.

After unloading the nodules from the mixer (a process which has been mechanised), they are dried and baked.

The most extensively used method for treating fine and dustlike iron ores is agglomeration (sintering) on the grate bars of high-output agglomerating machines handling from 2 to 2.5 thousand tons of ore per 24-hour day.

The special mixture prepared for agglomeration consists of ore fines (not larger than 5 to 8 mm), flue dust, concentrate obtained by magnetic concentration and coke breeze (a coke production waste) crushed to 3-mm lumps. By weight the coke breeze constitutes 6 to 10 per cent of the sintering mixture; the proportion of the fines, flue dust and concentrate is dictated by local conditions.

Before sintering, water is added to raise the moisture content to 5 or 6 per cent and the charge is thoroughly mixed in trommel or other types of mixers. In this process the finest moistened particles adhere and form pellets so that the whole mixture acquires a granular structure and becomes sufficiently permeable to gases. Thus conditioned, the mixture is fed to the grate of the sintering machine in a layer (bed) from 200 to 300 mm thick and is ignited by a gas nozzle from the top of the bed.

Powerful exhaust fans set up a vacuum of 600 to 1200 mm of water column in chambers (wind boxes) arranged below the grate. This vacuum effects air suction through the bed so that the flame front of the burning coke breeze, commencing at the top, proceeds downwards in a narrow strip through the bed and ends at the grate of the machine. The high temperature developed in the descending flame front (up to  $1450^{\circ}\text{C}$ ) leads to the sintering of the material in the bed and to the formation of the final porous product of the process which is called the agglomerate, or sinter.

It would be erroneous to think that sinter is produced as a result of the complete melting of the ore particles. It is formed due to the originating of the mineral fayalite ( $2\text{FeO}\cdot\text{SiO}_2$ ) in the burning zone having a melting point of about  $1210^{\circ}\text{C}$ . Fayalite, combining with other oxides of the mixture ( $\text{FeO}$ ,  $\text{SiO}_2$  and  $\text{CaO}$ ), produces even more fusible systems which can melt at  $1130^{\circ}\text{C}$ - $1200^{\circ}\text{C}$ . These systems soften and melt before the other components of the charge and act as a solvent and bonding material in relation to the less fusible or larger ore particles. The amount of fluid bond formed with the participation of fayalite depends upon the coke content of the mixture. The higher

the content, the more fayalite there will be in the sinter and the firmer the sinter will be after cooling.

A modern agglomerating (sintering) machine is shown schematically in Fig. 2. It consists of a frame 1 in which a continuous grate is arranged. This latter is made up of a definite number of pallets 2 which are metal trucks, not linked together, having wheels at the sides and travelling along an endless track in a vertical plane. The

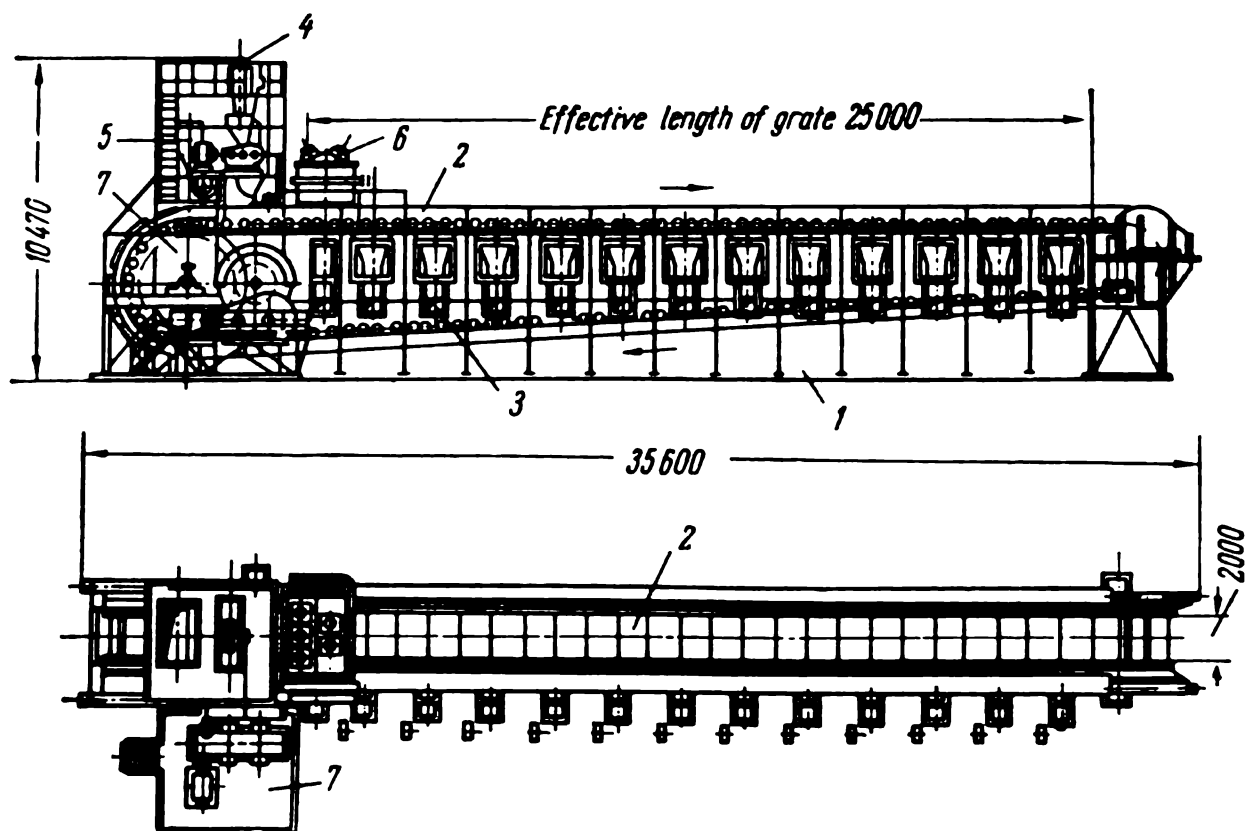


Fig. 2. Continuous-grate agglomerating (sintering) machine

pallets are picked up from the bottom, or return side, by a large driving sprocket 7 which then forces them to pass the length of the machine. Each pallet has only longitudinal sides and its bottom is formed by a grate. As the successive pallets push each other along the guide rails they form a long trough grate or hearth at the top of the machine in the horizontal section of the track. The wind boxes 3 are below this hearth.

The sintering mixture is loaded into feed hopper 4 at the head end of the machine which delivers it in a uniform layer on the grates of the pallets travelling along in a continuous line adjacent to each other.

A second hopper 5, at the head of the machine to the left of hopper 4, delivers the hearth layer on the free grates of the passing pallets. This thin layer consists of return fines of the sintering process; it protects the cast-iron grate against burning and also prevents the charge from falling through the spaces between the grate bars.

The coke in the mixture is ignited as the pallets pass under ignition furnace 6 which has two gas-burning nozzles. During the time each pallet travels along the horizontal section of the track all of the coke in the charge is consumed and ore sintering is completed.

When the pallet turns upside down as it goes over a circular guide at the discharge end of the machine, the hot sinter drops into a chute. The sinter is cooled and fines are screened out (sized) beyond the machine.

The sinter obtained in agglomeration has ample strength, excellent porosity and good reducibility; its use, therefore, improves blast-furnace operation by increasing the output and reducing to some extent the fuel consumption per ton of smelted pig iron.

*Fluxed sinter* has been produced on a wide scale by metallurgical plants in recent years. It is obtained by sintering iron ore with the addition of limestone. The optimum amount of limestone to add to the sintering mixture is such that the ratio of  $\text{CaO}$  to  $\text{SiO}_2$  in the finished sinter, i. e., the basicity of the latter, approaches the basicity of blast-furnace slag and, on the average, is within 1 to 1.4.

Before being fed onto the agglomerating grate the limestone is crushed into fractions up to 3 mm in size and is thoroughly mixed with the other components of the charge. In the agglomerating process the limestone is decomposed into  $\text{CaO}$  and  $\text{CO}_2$ ; the former participates in the formation of the sinter while  $\text{CO}_2$  is driven off with the gases. The reducibility of fluxed sinter is somewhat higher but its strength is lower than those of ordinary sinter.

Fluxed sinter facilitates slag formation in the blast furnace and enables the amount of limestone and coke charged into the furnace to be reduced. No flux whatsoever is charged when completely fluxed sinter is used.

## 2-5. Outline of the Blast-furnace Process

A flow diagram of blast-furnace operation is shown in Fig. 3.

*The raw materials* (ore, sinter, fluxes and coke) are delivered from the stockhouse by scale car 1 to the skip pit and loaded through hopper 2 into skip car 3. The loaded skip car is pulled up along the track of the incline, or skip bridge, by a steel hoist rope to the very top where it is dumped (position 3a). The charge falls into the receiving hopper of charging device 4 and from there into the lower hopper 5 closed by the large bell 5a. When the large bell is lowered the charge is admitted into the furnace proper (smelting room).

*The smelting room* of a modern blast furnace comprises a throat, stack, body (upper bosh line), bosh and hearth.

*The throat* is cylindrical in shape, it receives the charge and carries off the gases. *The stack* adjoining the throat is conical in shape

and tapers outward to the next lower section. This facilitates the descent of the materials being smelted and the distribution of the gases over the furnace cross section.

*The body* is the widest section; it is cylindrical in shape and connects the stack with the bosh. The *bosh* converges downwards to the hearth. This is done because here the volume of the materials being smelted is reduced and they are converted into the liquid products—pig iron and slag.

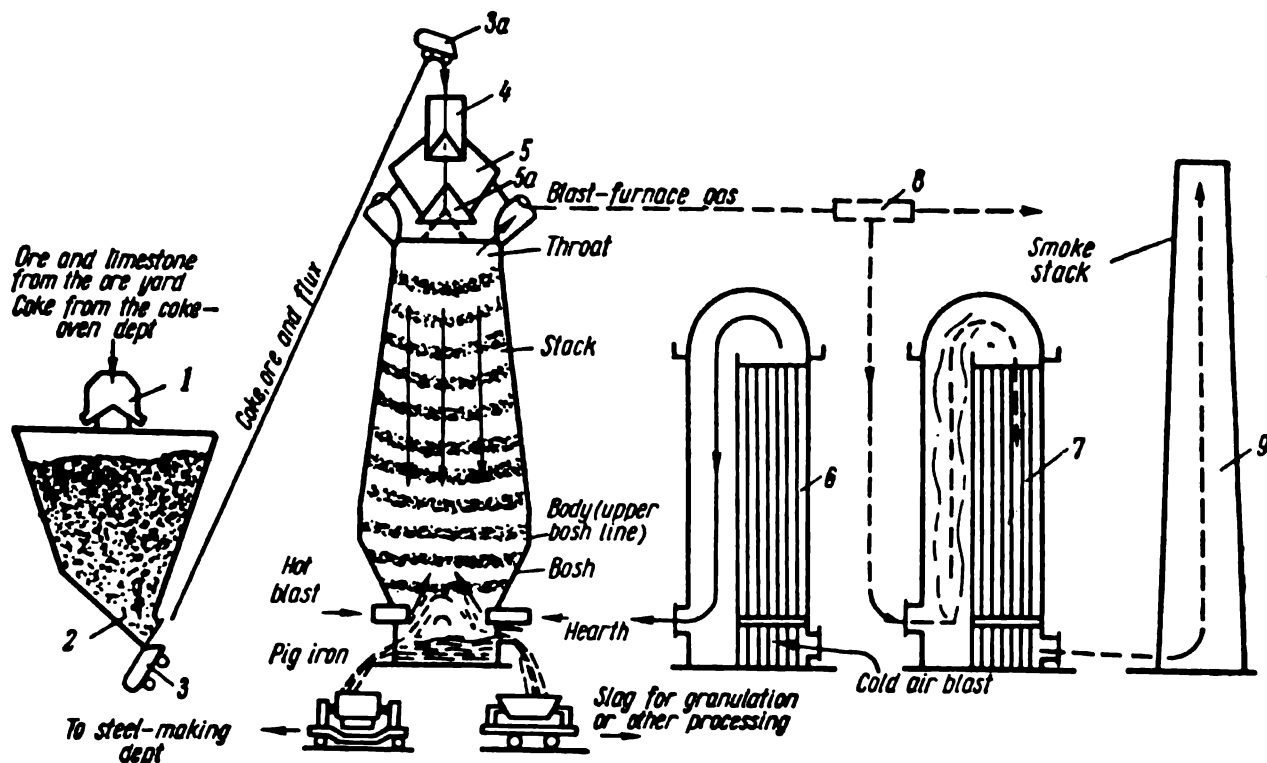


Fig. 3. Flow diagram of blast-furnace operation

The lowest part of the furnace, *the hearth*, is cylindrical in shape. The tuyeres, near which the coke burns, are spaced around the top of the hearth. The lower part of the hearth ends in *the bottom* on which the pig iron and slag collect before they are drawn off through the tapping hole and the cinder notch, or monkey, into iron and slag ladles.

A *hot blast* is forced into the furnace through the tuyeres to develop the high temperatures required and to intensify the smelting process. This is accomplished by passing the cold air blast from the blowers through the heated checkerwork of hot-blast stove 6.

While hot-blast stove 6 is giving up the heat stored in its brickwork to the cold air and cools down, a second stove 7 is being heated, i.e., it regenerates heat evolved in the combustion of blast-furnace gas, preliminarily cleaned in the dust catcher 8. The gaseous products of combustion evolved in hot-blast stove 6 are exhausted through

stack 9. At definite intervals, i.e., when the refractory checkerwork of stove 6 has cooled down and that of stove 7 has stored sufficient heat, the stove valves are reversed and the cold air blast passes into and through stove 7 while stove 6 is put "on gas" (it is heated).

In the many centuries that they have been used blast furnaces were subjected to many-sided improvements: their size has been greatly increased; they have acquired a more efficient shape (this refers to a vertical section through the smelting room); mechanically weak charcoal has been replaced by a stronger fuel, coke; preheated air blast has been applied, and operations involving the loading of the charge and disposal of the smelting products have been mechanised and automated. For a very long time the gases evolved in the smelting process and containing a large amount of carbon monoxide were merely discharged to the atmosphere. At present, they are collected and utilised for preheating the air blast, firing coke ovens and for other purposes.

A modern blast furnace is a very large structure (its effective height, for example, reaches 30 m and its volume is 1719 cu m and more) and has a high output—up to 2500 tons of pig iron per 24-hour day.

Modern blast furnaces consume large amounts of charging materials and air; they require highly organised work of all the sections of the blast-furnace department. It may be pointed out, for instance, that to produce 100 tons of pig iron it is necessary to charge the furnace with 180 tons of iron ore prepared for smelting (including sinter), 95 tons of screened coke and 50 tons of crushed and classified limestone, and to admit about 350 tons of preheated air blast. In addition to the pig iron, about 60 tons of slag and 500 tons of blast-furnace gas is produced.

## 2-6. Design of the Blast Furnace

**Furnace construction.** Fig. 4 is a cross section of a typical blast furnace, as well as its adjacent auxiliaries and mechanisms for charging the raw materials and for disposal and cleaning of the gases.

A blast furnace, consisting of the metallic shell, refractory masonry and the furnace top, rests on a very heavy and powerful reinforced concrete foundation 1.

The hearth bottom 3, consisting of several courses of high-quality fireclay brick or graphite-clay blocks, is laid within a recess at the top of the foundation. Steel columns 2 which support the mantle ring of the furnace stack are mounted on a reinforced concrete ring at the outer circumference of the foundation. The sidewalls of the hearth 4, bosh 5, body 6, stack 7 and throat 8 are lined with fireclay brick.

The refractory lining of the throat and stack are surrounded by a steel shell through which box-type coolers with circulating water are



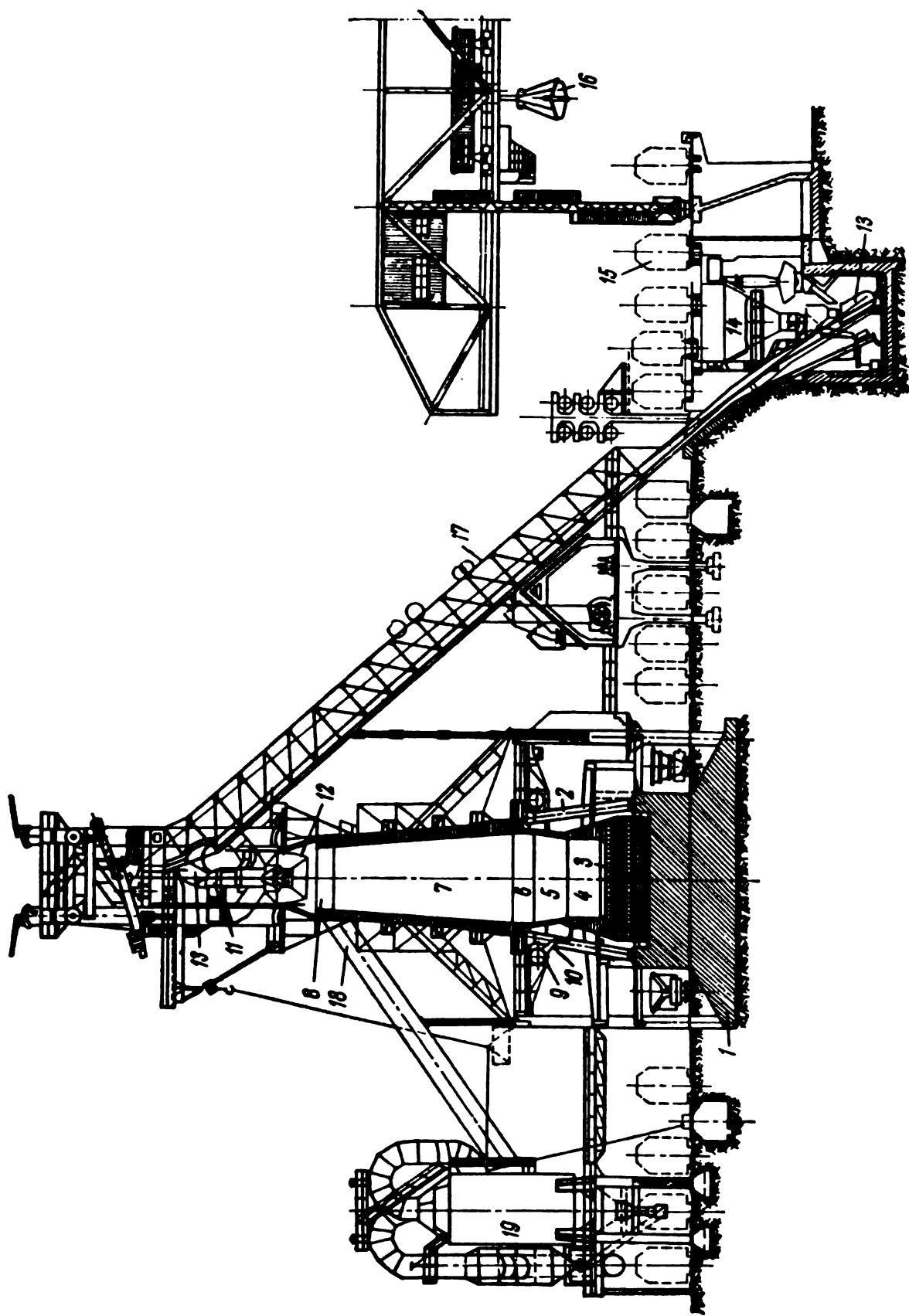


Fig. 4. Cross section of a modern blast furnace and its adjacent auxiliaries

passed. The sidewalls of the hearth bottom, hearth and bosh are enclosed in a steel jacket, or casing, with plate coolers.

The tapping hole (iron notch) is from 400 to 500 mm above the hearth level and, therefore, a stationary layer of liquid pig iron covers the bottom and protects it against destruction. The iron notch is in the form of a channel passing through the refractory masonry of the hearth. Between tappings this channel is plugged with refractory clay.

The cinder notch (monkey) through which the slag is drawn off is 1.5 to 1.6 m above the hearth level. The notch is a water-cooled, tapered, annular copper box, or tuyere, with the smaller end facing into the furnace and the larger, external end facing the cinder trough for slag removal. This notch is arranged inside the tapered, copper, intermediate cooler which, in turn, is inserted telescopically into a cast-iron notch cooler with coils for water circulation. The central hole in the notch is closed in the periods between slag tapping by a metal bott which is mechanically operated by the cinder-notch stopper.

Hot blast from the stoves (not shown in Fig. 4) passes through the hot blast main to the annular bustle pipe 9 surrounding the furnace at the level of the bosh. From the bustle pipe the hot blast enters each of the furnace tuyeres through the gooseneck 10 and a metal blowpipe. The design of the tuyeres is similar to that of the cinder notch. They consist of the water-cooled, copper tuyere proper, tapered bronze tuyere cooler and cast-iron jacket plate attached to the refractory masonry of the furnace. The water-cooled copper tuyere projects 150 to 200 mm inside the furnace to aid in diverting the blast away from the furnace walls. The exit hole of the tuyere is about 150-160 mm in diameter.

The furnace top mechanism above the throat consists of two independently operated units: (1) a narrow, rotating receiving hopper 11 which is closed by a small (upper) bell and (2) a large stationary charging hopper 12 which is closed by a large (lower) bell. The charged materials drop first onto the small bell since it is above the large bell.

After skip car 13 dumps a charge of the mixture into the receiving hopper the latter, together with the small bell, turns 60° about the vertical axis to help distribute the charge evenly over the surface of the large bell before it is lowered to admit the charge into the furnace.

The large and small bells of the charging machinery are controlled independently of each other. For this purpose the large bell is supported by a solid tie rod which passes through the hollow tie rod supporting the small bell. Both bell rods are linked to counterweighted beams which raise and lower the bells.

The charging materials are loaded into the skip cars from bins 14 which are filled either by transfer cars 15 or by grab bucket 16 from

the stock storage. The inclined skip bridge 17 serves to hoist the raw materials to the furnace top. The blast-furnace gases pass through the downcomers 18 into the dry dust catchers 19.

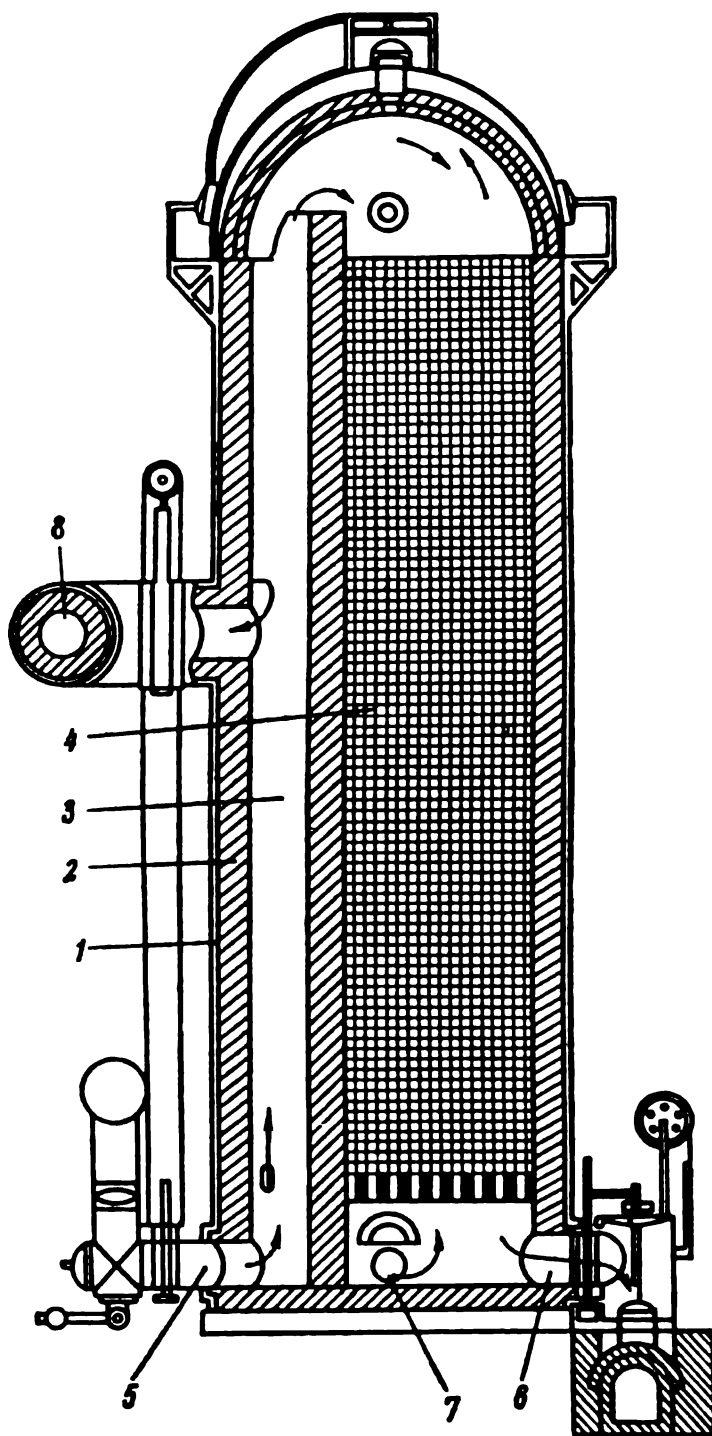


Fig. 5. Cross section of a hot-blast stove

**Obtaining the hot blast.** The air blast for the blast-furnace process is produced by powerful turbo-blowers or steam-turbine-driven blowers with a capacity up to 4000-4300 cu m per minute. The cold air blast is delivered by the blowers at a pressure up to 2-2.6 atm to the hot-blast stoves where it is preheated to 700°-950° C (depending

upon the grade of pig iron being smelted and the grade of coke being used).

The principle of a hot-blast stove of modern design is illustrated in Fig. 5. It has a steel shell 1 of cylindrical shape which ends in a dome. The lining (brickwork) of the wall 2 is made of high-grade fireclay brick.

The internal space of the stove is divided into two parts, namely, the combustion chamber 3 and the compartment with the refractory checkerwork 4. Cleaned blast-furnace gas is fed to burner 5 for firing the stove. Simultaneously, the required amount of air is delivered to the burner by a fan from the atmosphere. The air-gas mixture is ignited in the combustion chamber but its combustion is incomplete at this stage. It is completed in the dome of the stove from where the hot combustion products pass downwards through the checkerwork, to which they give up their heat, and are discharged through chimney valve 6.

When the refractory checkerwork of the stove has been heated to a temperature of  $1100^{\circ}$ - $1200^{\circ}$  C, gas and air feed to the combustion chamber is shut off and cold air from the turbo-blower is admitted through duct 7. The blast first enters the space under the checkerwork and then passes upward through the passageways, being gradually heated. The blast, heated to a temperature of  $800^{\circ}$ - $900^{\circ}$  C, passes through main 8 to the blast-furnace tuyeres.

The dimensions of the hot-blast stove, and the volume and surface area of the checkerwork it contains depend upon the working volume of the blast furnace. It may be assumed that 50 sq m of heating surface in the stove are required for each ton of blast-furnace output per 24 hours.

Along the height of the stove the checkerwork usually comprises three stories with passageways of various cross section for the hot gases and air blast.

## 2-7. Blast-furnace Operation

**Blowing in a blast furnace.** After erecting a new blast furnace and checking the operation of all its equipment, the furnace is blown in, i.e., prepared for 4.5 to 5 years of continuous operation (the active campaign of a blast furnace). Blowing in begins with slow drying of the refractory lining for 5 or 6 days.

When the lining is sufficiently dried and heated to some degree, the furnace is filled with the blow-in burden, the coke is ignited near the tuyeres and then the blast is turned on. The blow-in burden first consists of coke only (the bed charge) and further on of coke and a small amount of flux and ore. As the combustion of the coke progresses the composition of the charge is changed towards an increase in the

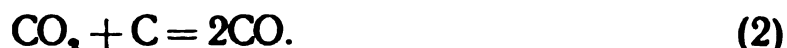
ore and flux content up to the specified limits. Slag is flushed for the first time from 15 to 20 hours after blowing in and pig iron is tapped after 24 hours.

**Physicochemical processes that take place in the blast furnace.** The working burden, i.e., a definite amount of coke, ore (sinter) and fluxes, is charged into the blast furnace in separate layers and in this form it participates in the smelting process. The smelting space of the furnace is divided between coke, ore and fluxes in the proportion 10 : 3.6 : 1.

In the course of coke combustion near the tuyeres the melting materials move downwards gradually, meeting the upward stream of hot gases evolved in the hearth and partly in the bosh of the furnace. Though the hot gases pass through the furnace in a very short time (5 or 6 seconds) they heat the column of charged materials to high temperatures which lead to corresponding physical and chemical changes in the fuel, ores and fluxes. At the furnace throat the temperature of the rising gases equals 300° to 350°C and it gradually increases at lower levels. The highest temperatures (1700°-1750° C) are developed at the upper part of the hearth in the vicinity of the tuyeres since it is here that the carbon in the coke burns according to the reaction



A large quantity of heat is evolved in this reaction and carbon dioxide ( $\text{CO}_2$ ) is formed which is almost completely decomposed upon contact with the hot coke:



Thus, the gaseous phase formed in the hearth and lower part of the bosh acquires intensive reducing capacity.

Upon contact with the rising gases the charged materials first lose their hygroscopic and then their chemically combined water. Due to their moisture loss (dehydration) the lumps of the charge decrease in weight, become more porous and sometimes crack. The combined (hydrated) water is driven off beginning with 102°-105° C and in some cases this process is completed only at 450°-500°C (e.g., from lumps of clayey matter— $\text{Al}_2\text{O}_3 \cdot 2\text{SiO}_2 \cdot 2\text{H}_2\text{O}$ ). The hydrated iron oxide ( $n\text{Fe}_2\text{O}_3 \cdot m\text{H}_2\text{O}$ ) in hematite ore is converted into ferric oxide ( $\text{Fe}_2\text{O}_3$ ) when the ore is heated to 200°-250°C. Dehydration of the charged materials begins in the furnace throat and is usually finished in the upper half of the stack. Here, the remaining volatile matter is removed from the coke (as  $\text{H}_2$ ,  $\text{CH}_4$  and other gases).

Thermal decomposition (dissociation) of the carbonates in the flux and in certain kinds of iron ores (siderite) occurs in the middle and lower sections of the stack. The temperatures at the beginning and

completion of this decomposition depend upon the nature of the carbonates and the size of the lumps; for example, the decomposition of limestone to form lime, according to the reaction



takes place at 900°-1000° C. Siderite decomposes to form magnetite by the reaction



at lower temperatures (400°-550°C).

The carbon dioxide evolved in the decomposition of  $\text{CaCO}_3$ ,  $\text{FeCO}_3$ , and other carbonates dilutes the blast-furnace gas, decreasing its carbon monoxide concentration.

When their moisture has been driven off, the lumps of iron ore and sinter undergo reduction which leads to the formation of metallic iron.

The reducing agents in the blast furnace are: (1) carbon monoxide (evolved near the furnace tuyeres in the combustion of coke), (2) hydrogen (evolved in the lower levels of the furnace upon interaction between the moisture in the blast and the carbon in the coke by the reaction  $\text{C} + \text{H}_2\text{O}_{\text{vapour}} = \text{CO} + \text{H}_2$ ) and (3) solid carbon (in the incandescent coke).

There is, as a rule, only a small quantity of hydrogen in blast-furnace gases and, therefore, the predominant proportion of the iron ore is reduced by carbon monoxide and solid carbon. Reduction by means of carbon monoxide begins in the stack and proceeds in steps from the highest iron oxide ( $\text{Fe}_2\text{O}_3$ ) to the intermediate oxide ( $\text{Fe}_3\text{O}_4$ ) and from the intermediate to the lowest oxide ( $\text{FeO}$ ) in accordance with the reactions

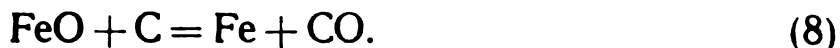


From a practical point of view the most important of the foregoing is reaction (7) which produces metallic iron as its end product. It is called the *indirect reduction* of iron and proceeds at moderate temperatures (500° to 900°C) with the evolution of heat.

In the presence of incandescent coke and at higher temperatures (above 1000°-1100°C), not only is iron ore reduced to metal in the blast furnace, but the regeneration of carbon monoxide proceeds very rapidly. This process may be expressed by the reactions

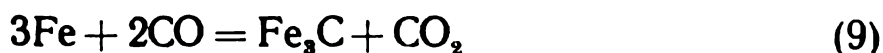


The simultaneity of reactions (7) and (2) enables them to be summed to obtain



The summary reaction (8) is called *the direct reduction* of iron. This reaction occurs because of the contact between the solid carbon of the coke and the ore or because of carbon deposited in the pores of the iron ore at low temperatures in the form of soot. It is assumed that about 60 to 50 per cent of the iron is produced in the blast furnace by reaction (7) and about 40 to 50 per cent by reaction (8). Direct reduction takes place in the furnace body and the higher the temperature of the materials here, the more complete it is since reaction (8) is endothermic, i.e., it absorbs heat.

Metallic iron formed in the blast furnace is initially in the solid state (spongy iron) since it has a high melting point (1539°C). In the presence of carbon monoxide spongy metallic iron is gradually carburised by the reaction



and its melting point is lowered to 1150°-1200° C. The result is that the carburised iron (containing up to 1.8-2%C) melts and runs down in drops between the lumps of incandescent coke on the well of the furnace hearth.

During the time the drops of metal run down, the iron absorbs additional carbon to a concentration of about 3.5 to 4 per cent, i.e., to the usual carbon content of liquid pig iron.

The reduction of manganese, silicon and phosphorus from the charge takes place in the blast furnace simultaneously with the reduction and carburisation of iron. These elements also go over into the pig iron.

The higher and intermediate oxides of manganese are reduced to the lower oxide in steps by carbon monoxide according to the sequence  $\text{MnO}_2 \rightarrow \text{Mn}_2\text{O}_3 \rightarrow \text{Mn}_3\text{O}_4 \rightarrow \text{MnO}$ . Manganous oxide MnO, the least reducible lower oxide of manganese, is reduced by solid carbon according to the reaction



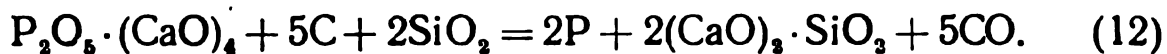
Reaction (10) is accompanied by the absorption of heat and occurs at temperatures above 1100°-1200°C. This circumstance dictates the procedures of smelting manganese cast irons; the blast furnace must consume more coke and be supplied with a hotter blast (800°-900°C). Silica ( $\text{SiO}_2$ ) is reduced in the blast furnace only by solid carbon according to the reaction



which also proceeds with heat absorption. In its pure form reaction (11) develops at 1450° C, but in the presence of reduced metallic iron

it begins at a lower temperature (1050° to 1100°C). Thus, in the smelting of siliceous cast irons, it is also necessary that the blast furnace run hot—with an increased coke consumption and with higher pre-heating of the air blast.

Phosphorus is introduced into the furnace charge in the form of phosphoric calcium salts and it is reduced in the presence of the iron ore gangue ( $\text{SiO}_2$ ) by solid carbon:



Reaction (12) proceeds easily in the blast furnace and as a result all of the phosphorus passes into the pig iron.

Sulphur contained in the charge behaves somewhat differently in the blast-furnace process than phosphorus does. A certain amount of sulphur is removed with the blast-furnace gases as  $\text{H}_2\text{S}$  and  $\text{SO}_2$ , but the main part remains in the blast furnace and is distributed in the form of sulphides between the liquid slag being formed and the liquid pig iron. Iron sulphide  $\text{FeS}$ , having a high solubility in metal, should be considered the most undesirable of the sulphur compounds.

If the slag is sufficiently saturated with lime (45 to 50 per cent) the following reaction is observed in the furnace hearth:



as a result of which part of the sulphur goes into the slag in the form of calcium sulphide which is insoluble in metal.

The process of slag formation must be controlled in the blast furnace if reaction (13) is to be more completely developed. In particular, the computed amount of limestone in the mixture should be strictly observed and a sufficiently high temperature should be maintained in the hearth. The latter depends upon the coke consumption, blast temperature and proper slag formation in the upper parts of the furnace.

Slag forms in the blast furnace in two stages. Primary slag forms approximately at the level of the furnace body or the lower part of the stack on the basis of a fusible mixture of several oxides—lime, silica, alumina and iron oxides. At a certain proportion of the foregoing components, the first portions of the liquid ferrous slag are formed at 1160°-1200° C. As it runs down into the hearth the primary slag is heated to higher temperatures and its chemical composition is changed since it dissolves coke ash, fluxes and the remainder of the ore gangue. Very little ferrous oxide remains in the final blast-furnace slag but it is enriched in lime, magnesium oxide and sometimes alumina. This circumstance sharply increases the desulphurising ability of the slag in the furnace hearth, thereby favourably affecting reaction (13).



## 2-8. Products of the Blast Furnace

Pig iron, slag and blast-furnace gas are the products obtained in the blast-furnace process.

The following kinds of *pig iron* are smelted in blast furnaces:

*Conversion pig*, which is used to make steel. According to the steel-making process applied and the stipulated requirements, this kind of pig iron is classified into: (1) open-hearth pig, grades M-1 and M-2, obtained in a moderately hot run of the blast furnace and containing 1.5 to 2.5% Mn, 0.4 to 1.5% Si, 0.15 to 0.30% P and 0.03 to 0.07 S; (2) Bessemer pig, grades B-1 and B-2, obtained in a hot run of the blast furnace from low-phosphorus ores and containing 0.6 to 1.5% Mn, 0.9 to 2% Si, up to 0.07% P and up to 0.06% S; and (3) Thomas pig, grade T-1, smelted from phosphorus-rich ores in a moderately hot run of the blast furnace and containing 1.6 to 2% P, about 1.4% Mn, up to 0.6% Si and up to 0.08% S.

*Foundry pig*, grades ЛК-00, ЛК-0, ЛК-1, etc., used in the iron foundry for making castings of various kinds, is smelted in a hot run of the blast furnace from a charge with a sufficiently high silica content. Foundry grades of pig iron contain from 1.25 to 4.25% Si and up to 1.3% Mn.

*Special pig irons*, or blast-furnace ferroalloys (ferrosilicon containing 10 to 12% Si; poor ferromanganese with 10 to 12% Mn; standard ferromanganese with 70 to 75% Mn and other alloys), which are used in deoxidising and alloying steel.

*The final composition of blast-furnace slag* is directly dependent upon the composition of the charge and the grade of pig iron being smelted. When conversion or foundry pig is produced in coke-fuelled blast furnaces, the basicity of the slag, i.e., the ratio  $\beta = \frac{\text{CaO} + \text{MgO}}{\text{SiO}_2}$  is selected in excess of unity (usually  $\beta = 1.2$  to  $1.4$ ). The chemical composition of final slags from coke-fuelled blast furnaces is within the following limits: 33 to 40%  $\text{SiO}_2$ , 42 to 48%  $\text{CaO} + \text{MgO}$  and 10 to 20%  $\text{Al}_2\text{O}_3$ . Slags of charcoal-fuelled blast furnaces are more acid, i.e., they contain more  $\text{SiO}_2$ .

Blast-furnace slags may be utilised in the production of cement and other building materials (slag building blocks, mineral wool, blocks for paving roads, etc.).

## 2-9. Efficiency of Blast-furnace Operation

The most important economic factors in the blast-furnace process are

(1) utilisation coefficient of the effective volume in the blast furnace;

(2) coke consumption per ton of smelted pig iron.

The *effective-volume utilisation factor* of a blast furnace is the ratio of the smelting room volume (effective-furnace volume) to the 24-hour output of the furnace:

$$K = \frac{V}{Q} \text{ cu m per ton,} \quad (14)$$

where  $V$  = effective furnace volume, in cu m,

$Q$  = pig iron output per 24 hours, in tons.

The less this factor, the better a blast furnace is considered to operate since a lesser volume of the smelting room is employed for each ton of pig iron smelted per 24 hours.

The numerical value of  $K$  depends mainly upon three factors, namely: (1) the kind of pig iron being smelted, (2) the gangue content of the iron ore and (3) the charge preparation. Upon very efficient operation of a blast furnace in the production of conversion pig,  $K = 0.65$  to  $0.68$ .

The *coke consumption* for the production of one ton of pig iron may range from 0.8 to 1 ton for open-hearth pig, 1.05 to 1.1 tons for Bessemer pig, 1.1 to 1.5 tons for foundry pig and 1.75 to 2.5 tons for special pig irons.

Extensive research has been conducted in the U.S.S.R. in recent years with the aim of intensifying the blast-furnace process and improving smelting efficiency. Great attention was given to problems concerning the preparation of the raw materials, to such concentration techniques as ore mixing to equalise the composition, ore sorting and sizing, washing coal before coking to improve coke quality, substituting sinter for iron ores, and the production of self-fluxing and fluxed sinters which substantially reduce the amount of limestone in the furnace charge.

Blast humidification and oxygen enrichment of the blast prove to be important measures for increasing blast-furnace output. It has been recognised that the use of a blast with increased (but constant) moisture content can raise the pig iron output of the furnace by 8 to 10 per cent. Enrichment of the blast with oxygen raises the temperature in the combustion zone of the furnace and reduces the nitrogen content of the furnace gases.

At the present time, up to 95 per cent of the pig iron produced in the Soviet Union is smelted in blast furnaces equipped with automatic blast temperature control facilities. This feature provides for more uniform smelting operation, reduces blast-furnace gas consumption for preheating the blast and increases the service life of the stove lining.

### Chapter 3

#### STEEL-MAKING PROCESSES

A comparison of the chemical compositions of pig iron and steel shows that the former contains more carbon, silicon, manganese, phosphorus and sulphur. Consequently, steelmaking involves processes which reduce the concentration of a number of elements contained in pig iron. This is achieved by oxidising smelting.

Three types of smelting equipment are employed for steelmaking in modern industry, namely: converters, open-hearth furnaces and electric furnaces.

Steel may be made of: (1) only molten pig iron—in the Bessemer and Thomas processes; (2) a combination of molten or solid pig iron and steel scrap—in open-hearth and electric furnaces and (3) only steel scrap—in electric furnaces.

#### 3-1. Making Steel in Converters

**Principle of a converter plant.** A converter for making steel by blowing air through molten pig iron is a pear-shaped vessel (Fig. 6). The converter has a shell 1, welded or riveted of steel plates, and lined with a refractory material 2. On the outside the lined shell is embraced by the steel trunnion ring 8. The converter is supported by trunnions 6 and 9 in uprights 10. The hydraulic tipping device 11 (in other cases—a reducing gear) serves to tilt the converter into its horizontal and vertical positions. The device is linked through a rack with a pinion mounted on trunnion 9.

The converter bottom is pierced with tuyeres 4 through which the air blast passes into the metal bath. The wind box 5, secured to the bottom, receives the compressed air and distributes it to the tuyeres. The blast enters wind box 5 through the gooseneck, or elbow pipe, 3 and hollow trunnion 6 which is linked through a revolving joint with connection 7.

Before pouring in the molten pig iron, the converter is turned on its trunnions to the horizontal position. In this position the molten metal is contained in the belly of the converter and does not reach the tuyeres in the bottom. The pig iron is poured through the mouth (nose opening) 12 of the vessel which is then slowly raised to the vertical position, bottom downwards, after starting the blast, to prevent the molten metal from running through the tuyeres into the wind box. The amount of pig iron poured into the converter must not exceed in volume one-fifth of the cylindrical part of the vessel. The rest of the

converter volume is required for metal circulation during the blowing period.

The type of refractory lining used in the converter depends upon the character of the steel-making process. In the Bessemer (acid) process the lining is of dinas brick or quartzite; a basic material (burnt dolomite) is employed for the Thomas (basic Bessemer) process. The linings of acid converters can withstand from 1000 to 2000

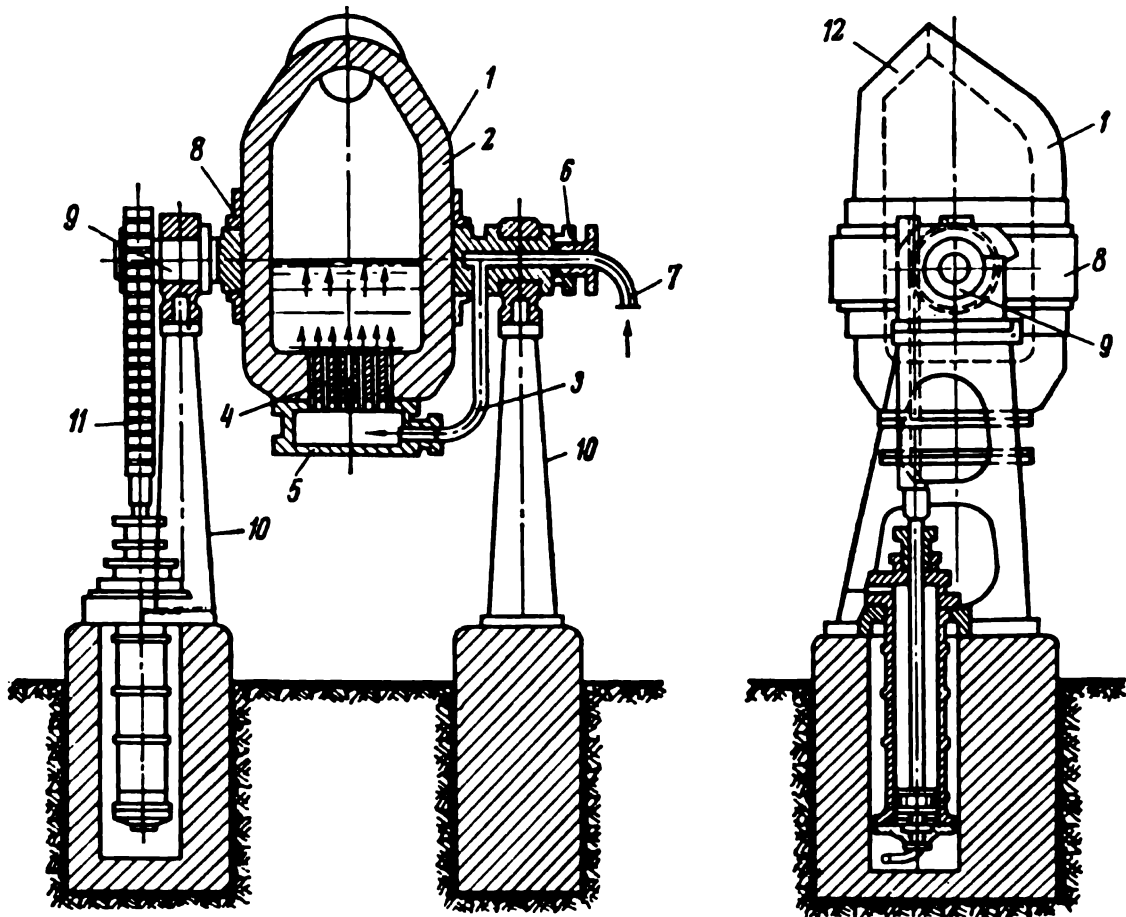


Fig. 6. Principle of the steel-making converter

runs; those of basic converters—from 350 to 400 runs. The bottoms of both types deteriorate more rapidly and are therefore more frequently changed.

Acid converters usually have a capacity of from 10 to 30 tons; the capacity of basic converters ranges from 25 to 40 tons and higher. Converters with a capacity of 60 to 80 tons and even higher are to be installed in new Soviet plants.

Air blast pressures in converters range from 2 to 2.5 atm; about 350 cu m of air is required per ton of pig iron. The number of tuyeres in the bottom depends upon the size of the latter; each tuyere hole is from 10 to 20 mm in diameter.

**The Bessemer process.\*** Pig iron used in the Bessemer process (in an acid-lined converter) should have a sufficiently high silicon and manganese content (up to 2% Si and up to 1.5% Mn), and the minimum possible amounts of sulphur and phosphorus. When the air blast is blown through the molten metal, reactions develop in which the iron, silicon and manganese burn to their oxides. These reactions proceed with intense evolution of heat so that the temperature of the metal bath is raised from 1250° to 1650° C.

*The slag-forming period (blow)* in converter operation begins immediately after the blast is turned on and continues for four or five minutes. The following reactions occur in this initial period of the process: when the very first portions of compressed air pass through the molten metal the iron is partially oxidised by the reaction



since about 92 to 94 per cent by weight of the charge poured into the converter is iron. A part of the ferrous oxide formed in reaction (15) goes into the slag but another part is dissolved in the molten metal and oxidises the silicon and manganese:



Reactions (15), (16) and (17) are accompanied by the evolution of a large amount of heat which raises the temperature of the metal as mentioned above. Together with the ferrous oxide, the oxides of silicon and manganese formed in these reactions provide an acid slag (40 to 50%  $\text{SiO}_2$ ). During the slag-forming blow, pellets of molten pig iron can be seen ejected by the blast from the nose opening; the pellets explode in the air and form sparks. This period is also called the silicon blow.

The second period of the Bessemer process—*the brilliant-flame blow* (also called the carbon blow)—begins after almost all of the silicon and manganese are burned out of the pig iron and the metal has reached a sufficiently high temperature. This creates favourable conditions for intensive burning of carbon from the molten bath:



Reaction (18) is accompanied by the absorption of heat and lowers the temperature of the metal by about 50° to 80° C. It takes about eight to twelve minutes, as a rule, to eliminate the carbon. The carbon monoxide evolved in reaction (18) rushes out of the converter

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\* First applied in 1856.

mouth and is oxidised by the air to produce a characteristic long brilliant flame above the converter.

The third, *reddish-smoke period*, is observed at the very end of the blow. It differs from the other periods in that a reddish smoke appears above the converter mouth which indicates that all of the impurities of the metal have been burned out and only iron is being oxidised. This period does not exceed one or two minutes after which the converter is turned to the horizontal position and additions of deoxidisers (ferromanganese, ferrosilicon or aluminium) are made to the metal bath to eliminate the oxygen and to bring the manganese and silicon content of the steel to the specified values.

Bessemer steel made by the process described above contains very little carbon since it was burned out in the second period.

Bessemer steel with a higher carbon content is obtained by two methods: (1) the blow is terminated at the moment when the molten metal still contains the given amount of carbon (young blown heats), and (2) the metal is decarburised to the end (full-blown heats) and then pig iron is added to the converter to raise the carbon content to the required value.

Metal losses in the converter range from 8 to 12 per cent.

Steel for foundry purposes is usually made in small acid-lined side-blown converters.

A *side-blown converter* (Fig. 7) comprises a steel shell 1 with a refractory lining 4 of dinas brick supported by trunnions on two up-rights 5. An air blast at a pressure of from 200 to 350 mm of mercury column enters tuyere block 3 through connection 2 and is then admitted into the converter through tuyeres in its side.

Before starting the blow the converter is slightly inclined so that the tuyeres make an angle of  $15^\circ$  to  $20^\circ$  with the surface of the molten metal. The blast rushing out of the tuyeres does not pass through the whole mass of the metal but only impinges on its surface. Part of the oxygen in the blast is used in oxidising the carbon monoxide evolving from the metal. This takes place within the converter and intensively heats the metal.

Molten pig iron for charging a side-blown converter is obtained in a cupola.

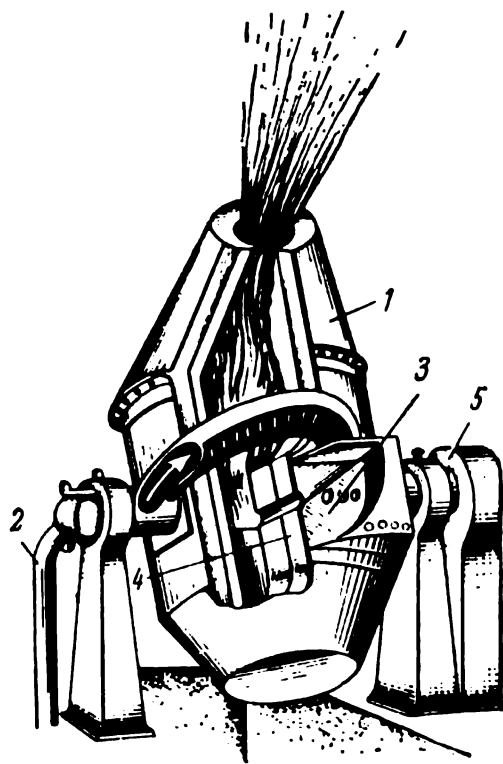
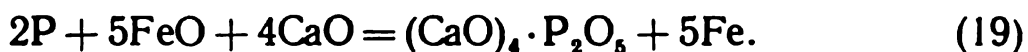


Fig. 7. A side-blown converter

**The Thomas process.** \* In the Thomas, or basic Bessemer, process steel is made in a converter with a basic lining (of dolomite brick). First a definite amount of freshly burned lime is charged into the converter, and then a high-phosphorus pig iron (1.6 to 2% P) with the least possible silicon and sulphur content (up to 0.6% Si and up to 0.07% S). In other words, the Thomas process is conducted with an artificial slag which is required to extract and combine the phosphorus in the pig iron.

The Thomas converter is charged and turned up, and the blast is started in the same manner as for the acid Bessemer process.

In the first period of the blow, iron, silicon and manganese are oxidised and a basic slag is formed (due to the lime addition). This raises the bath temperature somewhat but phosphorus is not yet removed from the metal. Carbon is burned out in the second period in which the temperature of the metal falls to some extent. When the carbon content in the metal is reduced to 0.1-0.2 per cent and the temperature drops to 1400°-1420° C, the run enters the third period. During the third period (from 3 to 5 minutes) phosphorus is intensively oxidised and slagged according to the reaction



A considerable amount of heat is evolved in reaction (19) and this leads to a new sharp increase in the temperature of the metal (up to 1600°C).

After finishing the third period of the blow, i. e., before adding deoxidising agents to the finished steel, it is necessary to remove the phosphorous slag by tilting the vessel. If this is not done the carbon, silicon and manganese contained in the deoxidisers reduce the phosphorus in the slag and transfer a part of it back into the metal. Thomas process slags contain up to 22%  $P_2O_5$  and are therefore used as fertilisers.

**Oxygen enrichment of the blast.** Enrichment of the blast with oxygen has been applied in recent years to speed up steel-making processes in converters.

In the acid Bessemer process oxygen enrichment reduces blowing time and raises converter output by as much as 15 to 20 per cent. The latter is achieved by increasing the proportion of steel scrap in the charge (up to 30 per cent instead of the usual 5 to 8 per cent). The main advantage, however, in using an oxygen-enriched blast in the acid Bessemer and Thomas processes is the appreciable reduction in the amount of nitrogen in the finished steel (down to 0.008-0.010 and even 0.004-0.005 per cent instead of 0.022-0.03 per cent obtained with an ordinary air blast).

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\* First applied in 1878.

The introduction of a mixture of oxygen and steam or carbon dioxide gas into the blast has proved to be especially effective. This procedure enables the quality of Bessemer steel to be raised to that of open-hearth and even electric furnace steel. The field of application of Bessemer and Thomas steels can be widened by using oxygen in converter processes.

### 3-2. The Open-hearth Process

Very high flame temperatures (up to  $1800^{\circ}\text{C}$ ) are achieved in open-hearth furnaces due to the fact that they operate on preheated fuel gas and air. Before entering the furnace the fuel and air for combustion are preheated in the checkerwork of the regenerators.

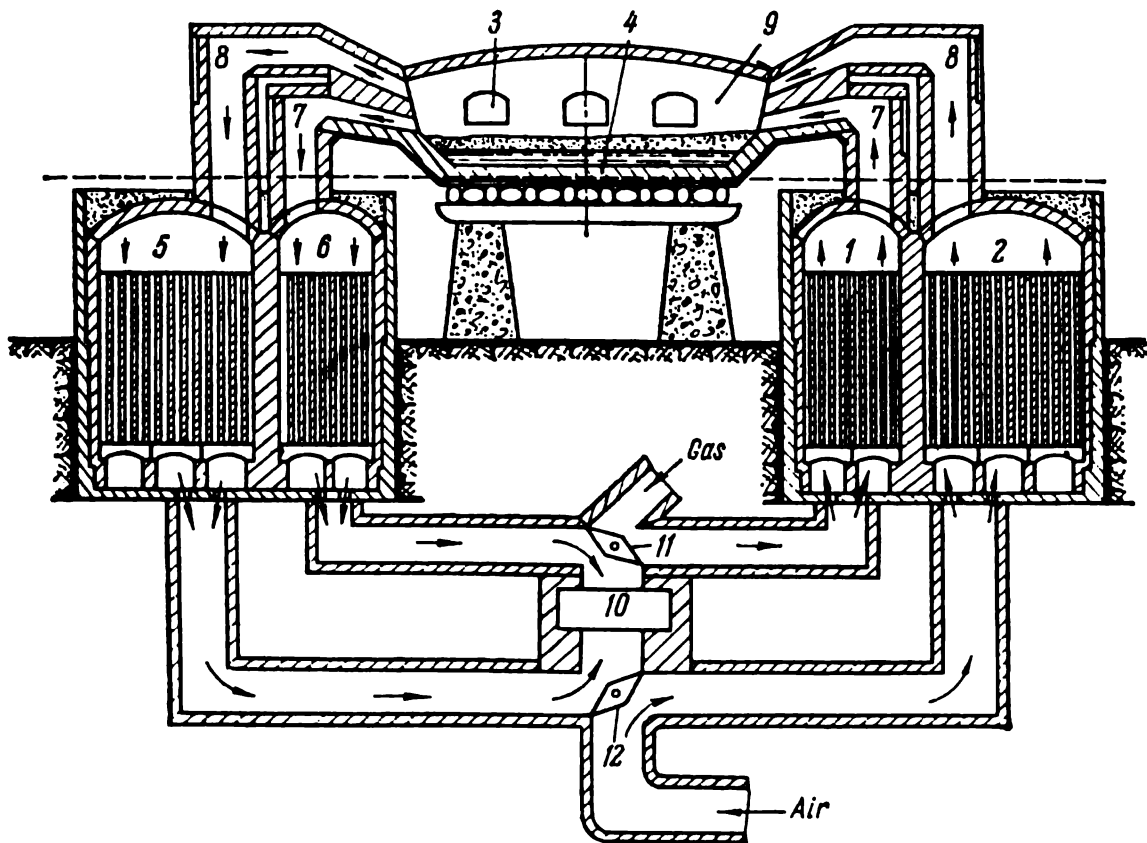


Fig. 8. Operation of an open-hearth furnace

Steelmaking in the open hearth comprises the following periods: (1) charging and melting down the charge, (2) boiling of the bath of molten metal and (3) refining and deoxidising.

Fig. 8 is a schematic diagram of an open-hearth furnace showing the direction of gas flow during operation of the right-hand blocks. Before entering the melting chamber of the furnace, the fuel gas and the air required for its combustion pass through the checkerwork of regenerators 1 and 2, previously heated to a temperature of about



1200° C. The gas and air, heated in the regenerators to about 1000°-1100°C, then pass up along the vertical flues of the right-hand blocks 7 and 8 and mix together in the melting chamber 9 where the combustion reaction takes place. This burning of the gas raises the temperature under the furnace roof to about 1700° C.

The products of combustion, or flue gases, pass from the melting chamber through the left-hand blocks 7 and 8 into the chambers of regenerators 5 and 6 where they heat the refractory checkerwork and then pass out through stack 10. When the checkerwork in chambers 1 and 2 becomes cool due to incoming gas and air, the flow of the latter is reversed by valves 11 and 12 and directed into chambers 5 and 6 which have by this time been heated to 1200°-1300° C by the flue gases. The latter, after reversal, pass through the checkers of chambers 1 and 2. Thus, a constant supply of preheated fuel gas and air is ensured to the melting chamber.

Most open-hearth furnaces are fired with a mixture of blast-furnace and coke-oven gases.

The charging materials (scrap, pig iron and fluxes) are placed on the hearth 4 of the furnace through the charging doors 3 (see Fig. 8). The charge is heated, and the metal and slag are melted and further heated by the heat of combustion of the fuel in the melting chamber upon contact of the materials with the hot gases. The finished metal is tapped through a taphole and spout located in the lowest part of the hearth. During smelting the taphole is plugged with refractory clay.

Modern open-hearth furnaces are built for various capacities and can accommodate a metal charge up to 500 tons or more.

The hearth of the furnace may be either acid or basic. This circumstance determines the character of the metallurgical processes that occur in the furnace and the quality of the finished metal.

In acid open-hearth furnaces the refractory lining of the hearth, walls and roof is of dinas brick. The parts of the furnace subject to the severest conditions—the upper layer, or bottom, and banks of the hearth on which the molten metal and slag rest—are rammed of silica sand and then fritted (sintered). This layer is repaired after each heat. Only acid slag with a high silica content (50 to 55%  $\text{SiO}_2$ ) may be present when steel is made in an acid open-hearth furnace.

In a basic open-hearth furnace the hearth and walls are lined with magnesite brick while the roof is of dinas or chrome-magnesite brick. The bottom and banks are burned in with grain magnesite or dolomite, and are repaired after each heat. In the basic furnace steel is smelted under a basic slag (40 to 50%  $\text{CaO}$ ).

Both basic and acid open-hearth processes are characterised by a number of distinct modifications, depending upon the choice of

initial materials which constitute the charge. In some plants, usually of the mechanical engineering type, a solid charge is used. This procedure is called the *scrap-and-pig process* and consists in charging solid pig iron, scrap and a small quantity of iron ore into the furnace at the beginning of the process.

In metallurgical plants with operating blast furnaces, open-hearth furnaces are charged with liquid pig iron (60 per cent), waste metal and scrap (40 per cent), iron ore and fluxes. This method is called the *scrap-and-ore process*.

**The basic open-hearth process.** The features of steelmaking in a basic open-hearth furnace operating on a solid charge (scrap-and-pig process) are considered in the following.

Before starting a heat, the initial materials (pig iron, steel scrap, limestone and iron ore) are chosen, their proportions in the charge are determined, as well as the order in which they are to be charged into the furnace. The basis for the choice of materials is that the carbon content of the metal after melting should be from 0.6 to 0.8 per cent more than required in the finished steel after tapping.

First the light scrap is charged, then about 30 per cent of the heavy scrap is added and almost all of the limestone and ore required to form the primary slag.

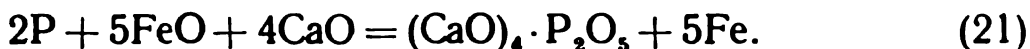
During the charging and melting-down stages in the furnace the iron and phosphorus are partially oxidised, the silicon and manganese are almost completely oxidised and the primary smelting slag is formed. At the beginning of the heat these elements are oxidised by the oxygen in the furnace gases and ore; after the formation of the primary slag, they are oxidised by the ferrous oxide dissolved in the slag.

When a layer of slag forms on the bath the liquid metal is isolated from direct contact with the gases and oxidation of impurities proceeds under the slag layer. Under these conditions the oxygen is supplied by the ferrous oxide ( $\text{FeO}$ ) which is dissolved in both the slag and the metal on the basis of the law of distribution. According to the latter an increase in the ferrous oxide concentration in the slag leads to an immediate increase in its concentration in the metal bath. This is why the melter is careful to maintain a predetermined ferrous oxide concentration in the slag and adds ore to the furnace at the required moment which leads to the reaction



The ferrous oxide, formed as a result of reaction (20), then passes from the slag into the metal and oxidises the impurities according to reactions (16), (17) and (18) that were taken up in the study of the Bessemer and Thomas processes.

At the end of the melting period in the basic open-hearth furnace, a considerable part of the phosphorus passes into the slag since the latter contains an ample quantity of ferrous oxide and lime. This promotes the development of reaction



To prevent the phosphorus from going back into the metal, about 40 to 50 per cent of the primary slag is removed (run off) before the beginning of the second period and lime is added at the same time to form a new and more basic slag. The fuel rate of the furnace is raised so that the lime, having a high melting point, is more rapidly dissolved in the remainder of the primary slag and so that the temperature of the metal bath will be as high as possible. A certain amount of iron ore is charged into the furnace 15 to 20 minutes after the more basic slag has formed. The ore, added in several portions, increases the iron oxide content of the slag and leads to reaction (18) in which the carbon in the metal bath is oxidised.

Carbon monoxide formed in reaction (18) is evolved from the metal in the form of bubbles thereby causing the metal to boil. Iron ore is usually charged into the furnace in the first half of the boiling period which is called the *ore boil*. The turbulence caused by the boiling of the bath facilitates more thorough mixing of the metal, better separation of nonmetallic particles and dissolved gases and also enables a more uniform distribution of the temperature to be obtained through the whole depth of the bath. Ample heat must be provided for proper boiling of the metallic bath since reaction (18) proceeds with an absorption of heat.

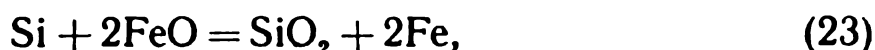
The time required for carbon oxidation depends upon the capacity of the open-hearth furnace and the grade of steel being made. This period may last from one hour and 15 minutes to two hours and 30 minutes and even more. The rate of carbon oxidation ranges from 0.4 to 0.65 per cent per hour.

No iron ore is added to the bath in the second half of the boiling period which is called the *pure boil*. The metal boils with small bubbles due to iron oxides collected in the slag. Here carbon is oxidised at a rate of 0.3 to 0.4 per cent per hour.

Control of the slag properties is of importance during the boiling period, particularly its fluidity and basicity.

When the carbon content in the boiling metal becomes somewhat lower than required for the finished steel (this is determined by proximate analysis of samples) boiling is artificially precluded and the heat enters its last stage—the *refining and deoxidising period*. For this purpose a predetermined amount of lump ferromanganese (containing 12% Mn) is charged into the open-hearth furnace and later (after 10 or 15 minutes)—ferrosilicon (with 12 to 16% Si). These

ferroalloys lead to the reactions



as a result of which the ferrous oxide content of the metal is decreased (the metal is deoxidised) and the carbon oxidation reaction is stopped. A visible indication of metal deoxidation is the discontinuance of the evolution of carbon monoxide bubbles from the surface of the slag. The slag becomes quiescent.

Sulphur is partially removed from the metal (desulphurisation) in a basic open-hearth furnace according to the reaction



Reaction (24) proceeds more effectively at high temperatures and with high basicity of the slag, in particular, it progresses actively at the end of the boiling period when the deoxidisers are added to the metal.

Many grades of alloy steel may be made in a basic open-hearth furnace. This is done by adding the required alloying elements—copper, chromium, nickel, etc.—or the corresponding ferroalloys—ferrochromium, ferrotitanium, ferromanganese and others—to the bath after deoxidising the metal.

Before tapping alloy steel it is held in the furnace for a certain time under a layer of slag to distribute the elements more uniformly through the volume of metal and to degasify the latter to some extent. Final deoxidation is accomplished by means of a spout or ladle addition of aluminium.

In the scrap-and-ore process, after properly heating up the solid charge (steel scrap, ore and limestone) on the furnace hearth, molten pig iron (up to 60 per cent weight of the metal charge) is charged through a runner at the charging door. Chemical reactions, and the removal of silicon, manganese, phosphorus, carbon and sulphur take place in the same manner as in the scrap-and-pig process.

**The acid open-hearth process.** The acid open-hearth process requires highly pure charging materials containing a minimum amount of phosphorus and sulphur (less than 0.03 per cent of each). This is due to the fact that the acid slag formed in the furnace is not capable of retaining these harmful impurities of steel.

Acid open-hearth furnaces usually operate on a solid charge; the proportion of scrap ranges from 30 to 50 per cent by weight of the metal charge. As a rule, the silicon content of the charge is held below 0.6 per cent since the boiling period starts later when the silicon content is high. No iron ore can be added to the charge since it can interact with the silica of the hearth lining and rapidly destroy the

bottom and banks, forming the fusible compound  $2\text{FeO} \cdot \text{SiO}_2$ . The primary smelting slag is formed in an acid open-hearth furnace by charging a certain amount of quartzite or, still better, acid open-hearth slag from previous heats.

After charging and heating the materials in the furnace, iron, silicon, and manganese are oxidised and fuse with the fluxes to form the acid primary smelting slag (up to 40%  $\text{SiO}_2$ ). In this slag a large part of the iron oxide is combined in the silicate form thus preventing the ferrous oxide from going over to the metal. Moreover, acid slags possess increased viscosity which affects the rate of carbon oxidation.

The metal boiling process is intensified in an acid open-hearth furnace by charging small portions of iron ore on the top of the slag (to increase the percentage of iron oxide in the slag), as well as burned lime (to fluidify the slag). As a rule, the primary smelting slag is not run off in an acid furnace since it contains no phosphorus.

There are certain distinctive features in the behaviour of silicon and silica in an acid furnace. Silicon is oxidised during the melting of the charge by the oxygen of the furnace gases:



and after the charge melts—by the ferrous oxide in the slag according to reaction (16).

Reaction (16) is reversible. At moderate temperatures it proceeds to the right while at high temperatures it may proceed in the opposite direction. This may be observed at the end of the heat when the temperature of the metal and the slag is raised substantially.

The following reactions are feasible at high temperatures and in the presence of sufficient amounts of carbon and manganese in an acid furnace:



The silica in the slag and bottom lining may participate in reactions (26) and (27), thereby taking an active part in the smelting process.

The silicon reduced from the slag or furnace bottom has a deoxidising effect on the steel and enables very dense metal to be obtained.

**Silicon-reduction open-hearth process.** An acid open-hearth process which is conducted with relatively high reduction of silicon from the slag and furnace bottom is called a *silicon-reduction process*.

In a well developed silicon-reduction process there is no need for adding deoxidisers to the metal bath; final deoxidation by a ladle addition of aluminium will prove sufficient.

A layer of acid (silica) slag with a low ferrous oxide content protects the metal against supersaturation with oxygen; therefore, at the

end of the process, i. e., before tapping the furnace, acid steel will contain less dissolved oxygen than basic steel.

Due to the high cost of the initial materials and the much longer boiling period, the acid open-hearth process is employed at present only for making high-quality grades of steel.

**Economical efficiency of open-hearth furnace operation.** The principal economic factors of open-hearth operation are:

- (1) fuel consumption per ton of steel made;
- (2) steel production in tons per square metre of hearth area per 24-hour day.

The fuel consumption per unit weight of liquid steel depends upon many factors, such as the composition of the charge, the thermal capacity of the furnace, etc. It has been established in practice that to obtain the required amount of heat (from 700 to 1400 kcal per kilogram of metal) it is necessary to burn reference fuel in a quantity from 10 to 20 per cent by weight of the steel being made. Minimum fuel consumption is observed in properly operating furnaces with a liquid charge while maximum consumption is characteristic of furnaces operating on a solid charge or furnaces with a poor thermal load due to the inexpedient design of the furnace blocks, ineffective regenerators, low roof durability, etc.

The steel production per square metre of furnace hearth area is calculated by dividing the daily (24-hour) output of the furnace (in tons) by the hearth area (in square metres) measured conventionally at the level of the charging door thresholds. Metallurgical plants of the Soviet Union produce from 7.6 to 8 tons of steel per square metre on an average; certain plants achieve a daily production of 9 to 10 tons per square metre. The development of high-speed steel-making methods promotes a further increase in furnace output and enables the average daily steel output per square metre of hearth area to reach 12 tons.

The most important method of intensifying the open-hearth process is oxygen enrichment of the air used for fuel combustion. This enables the flame temperature to be raised, the flame radiation capacity to be increased, and the amount of combustion products to be reduced. These factors increase the heating power of open-hearth furnaces. The use of oxygen in the fuel stream and the introduction of oxygen into the bath to eliminate carbon, increase the rate of production of open-hearth furnaces by 25 to 30 per cent.

Making the furnace roof of chrome-magnesite brick instead of dinas brick also increases the heating power of open-hearth furnaces; this measure prolongs the period between repairs by 2 or 3 times and increases the output by from 6 to 10 per cent.

At the present time 87 per cent of the open-hearth steel produced in the U.S.S.R. is made in furnaces equipped with automatic heating

control facilities. Such automatic control achieves fuel economy (about 5 per cent), increased output (8 per cent on an average) and prolonged roof life (by 9 per cent).

### 3-3. Electric Furnace Steelmaking

The most highly perfected steel-making units are electric furnaces in which electric energy is converted by various methods into thermal energy for heating and melting the metal.

Electric furnaces used in making steel are of two types—arc and induction (high frequency) furnaces. The first are more widely employed in metallurgy.

Electric furnace steelmaking possesses many advantages in comparison with open-hearth and other steel-making processes. It is possible in electric furnaces to (1) obtain very high temperatures (up to  $2000^{\circ}\text{C}$ ) so as to melt metals with a high concentration of components with high melting points (chromium, tungsten, molybdenum and other admixtures), (2) operate with highly basic slag (up to 55-60%  $\text{CaO}$ ) and to remove a large part of the phosphorus and sulphur from the metal, i.e., to effectively free the metal of harmful impurities, and (3) set up a reducing atmosphere or a vacuum (in induction furnaces) thereby achieving better deoxidation and degasification of the metal.

An electric-arc furnace for steelmaking is shown in Fig. 9. In this furnace the charge is heated and melted by heat radiated from three arcs—according to the number of phases of alternating current (only two electrodes are seen in Fig. 9; the third is in the cut-away part of the section). The electric arcs are struck in the melting chamber of the furnace between the vertically mounted electrodes and the metallic charge.

An electric-arc steel-making furnace comprises a cylindrical welded or riveted shell *1* with a spheroidal bottom; refractory lining *2* of the hearth and walls; removable dome roof *3* assembled of shaped refractory brick in a roof ring (holes are left in the roof lining for the electrodes *7*); mechanism *10* for supporting and feeding down the electrodes; two supporting toothed rockers *4* on which the furnace is held and tilted on the toothed rails *11* of the foundation; and the tilting mechanism *6* which enables the furnace to be tilted towards the tapping door and spout *5* for pouring.

Power supply of the furnace is from the step-down transformer *8* located in separate premises. Copper busbars and flexible cable *9* are used as power leads to the electrodes.

The roof of an electric-arc furnace is most frequently made of dinas brick but sometimes chrome-magnesite blocks are used for this purpose.

Carbon or graphitised electrodes are employed in electric arc furnaces. Graphitised electrodes offer less resistance to current and are considered to be more durable at high temperatures. It is more expedient to use graphitised electrodes for steelmaking but they are more expensive than the carbon type.

In the course of the heat the lower ends of the electrodes are consumed and the electrodes become shorter, therefore they are gradually

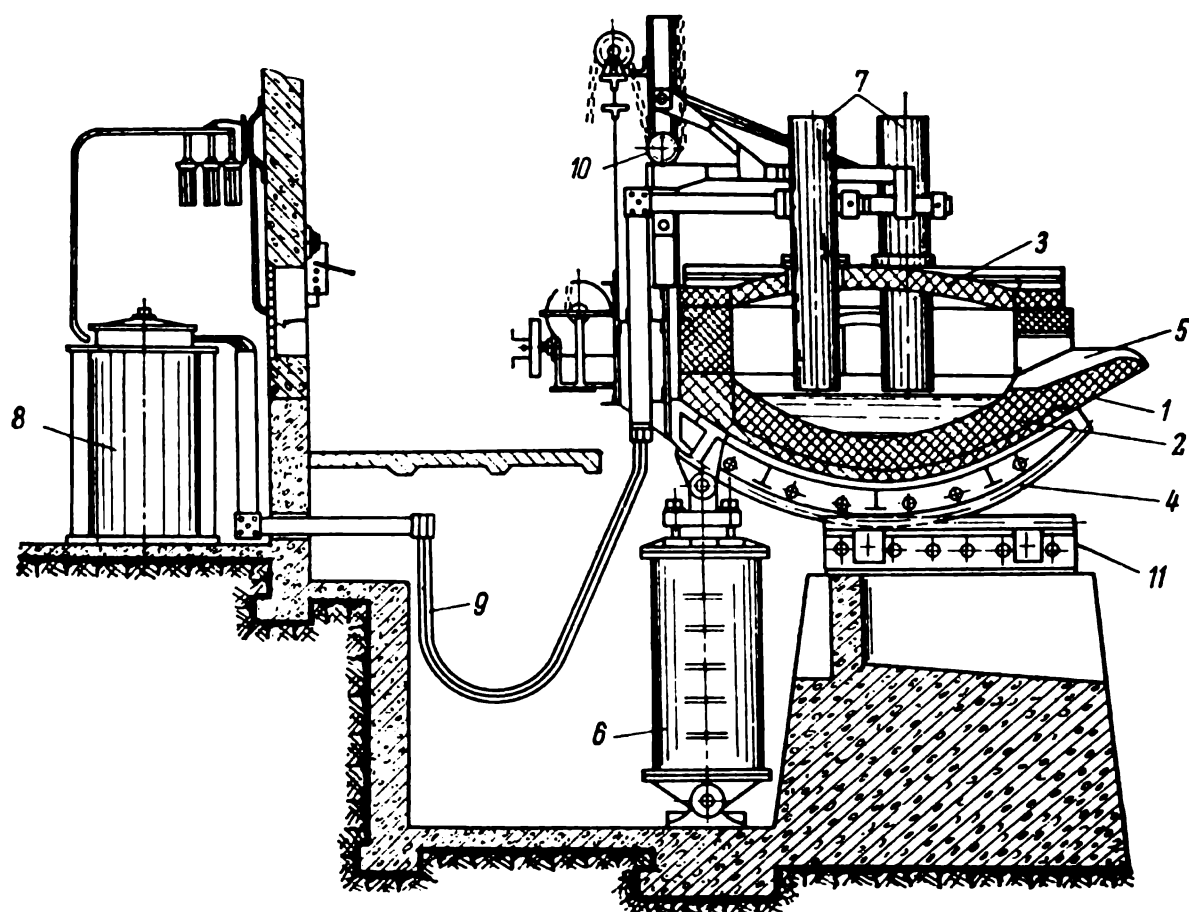


Fig. 9. Electric-arc furnace, 80-ton capacity

fed down into the furnace and, whenever necessary, a new electrode is built up (screwed) on top of the old one. Each of the three electrodes is held tightly in the contact cheeks of the spring clamp to which power is supplied from the secondary winding of the furnace transformer. The primary of this transformer is supplied with high-tension current—from 6000 to 30,000 V—this current being transformed to a lower voltage—from 90 to 280 V depending upon the selected voltage stage. The power rating of the transformer depends upon the capacity of the furnace and the particular steel-making technique being employed. The electric energy consumption of arc furnaces operating on a solid charge may be taken as 600 to 950 kWh per ton of finished steel.



The capacity of electric-arc furnaces ranges from 3 to 180 tons and even more.

**Steel-making procedures in electric-arc furnaces.** For the most part high-quality grades of carbon and alloy steels are made in electric-arc furnaces from appropriate charging materials. As a rule, such furnaces operate on a solid charge.

Arc furnaces with a basic lining operating on a solid charge produce steel by one of two principal methods: (1) with oxidation of the carbon and other components of the charge, and (2) without oxidation of the charge (remelting method).

*Steel-making procedure with oxidation of the charge* has much in common with the above-described scrap-and-pig process in an open-hearth furnace. After repairing the bottom a predetermined charge is put into the melting chamber of the furnace. The mean carbon content of the charge should be 0.5 to 0.6 per cent higher than that required in the finished steel. Surplus carbon is burned out of the molten metal and ensures proper boiling of the bath.

First, light scrap is charged on the furnace bottom, and then heavier pieces (crop ends of rolled stock, small ingots, etc.). The charge should be placed as densely as possible in the area within, or adjacent to, the triangle formed by the electrodes where the arc will be struck. Upon completion of the charging operation the electrodes are lowered from under the roof into light contact with the pieces of metal. First, lumps of coke are inserted under the ends of the electrodes to ensure smoother striking of the arcs, then the current from the furnace transformer is switched on and the heat begins.

Steelmaking in an electric-arc furnace is divided into the oxidising and reducing periods.

During *the oxidising period* the solid charge is melted down; silicon, manganese, phosphorus, surplus carbon, to some extent iron and certain alloying elements, if present in the charge (for example, chromium and titanium), are burned out of the metal. Separate elements are oxidised according to reactions (16) through (24). Phosphorus is removed from the metal in an electric furnace with a basic lining chiefly in the first half of the oxidising period, before the bath has been heated to a great degree. From 60 to 70 per cent of the phosphorous slag formed at this time is removed from the furnace and then freshly burned lime and other required components (iron ore, bauxite, fluorspar, broken chamotte brick, etc.) are charged to adjust the basicity of the primary smelting slag. After running off a large portion of the primary slag the metal is thoroughly heated and a new burden of slag-forming materials—iron ore or scale—is charged into the furnace. Soon afterwards, the liquid slag of the second half of the oxidising period is completely formed and the bath begins to boil.

During the boil, which lasts at least from 45 to 60 minutes, surplus carbon is burned out of the metal, dissolved gases are evolved and nonmetallic inclusions are removed. Then, samples of the metal are taken for a proximate analysis to determine the carbon and manganese content, and of the slag to determine the  $\text{CaO}$  and  $\text{SiO}_2$  content. The basicity of the slag, i.e., the lime-silica ratio, is maintained at from 2 to 2.5. This is necessary to enable the slag to retain the phosphorus and to provide a sufficient oxidising capacity when iron ore is added.

When the carbon is removed, all the slag of the second half of the oxidising period is run off. If the carbon content of the metal was lowered excessively during the boil, pieces of broken graphite electrodes or coke are added into the furnace to carburise the metal to the specified value.

Two methods are used to conduct the reduction in arc furnaces with a basic lining: (1) under a white (lime) slag for low-carbon steels and (2) under a carbide slag for high-carbon steels.

A white slag is obtained in the following manner. After removing the remainder of the slag formed in the oxidising period, the first slag-forming mixture of the reducing period is placed on the surface of the metal. This mixture amounts to 3 or 4 per cent of the metal by weight and consists of 80 per cent lime and 20 per cent fluorspar (fluorspar is added to reduce the melting point of the slag). Soon a layer of slag with a sufficiently high  $\text{FeO}$  and  $\text{MnO}$  content is formed on the surface of the metal. A solidified sample of this slag is of a dark colour; such a slag is not yet capable of removing oxygen in the form of  $\text{FeO}$  from the metal.

Before the metal begins to deoxidise, the second slag-forming mixture of the reducing period is charged into the furnace to lower the concentration of iron and manganese oxides in the layer of slag.

The second slag-forming mixture is ordinarily composed of well burned lime (4 or 5 parts), fluorspar (1 part) and ground charcoal and coke (2 or 3 parts). These components are thoroughly shovelled together and placed on the surface of the reducing-period slag in the furnace in two or three portions. After a certain time the composition of the furnace slag is altered in the direction of an increase in basicity and a decrease in the  $\text{FeO}$  and  $\text{MnO}$  content. Solidified samples of such slag become lighter in colour; the slag acquires the ability to deoxidise steel by diffusion, i.e., extracting dissolved  $\text{FeO}$  from the metal.

At the end of the reducing period ferrosilicon powder is added on the metal surface to intensify the deoxidising action of the white slag. As a result, the  $\text{FeO}$  content in the slag is lowered to about 1-1.5 per cent. Since the white slag contains from 50 to 60 per cent  $\text{CaO}$  and there are lumps of charcoal or coke on its surface, sulphur

is intensively removed from the metal according to reaction (13). The high temperatures required for reaction (13) are easily attainable in an electric-arc furnace.

During the reducing period the steel in the furnace is brought to the specified composition; all the required additions are made, including the alloying elements.

Final deoxidation of the steel is accomplished by means of aluminium.

Steelmaking under a carbide slag is conducted in the first stage of the reducing period (until solidified samples begin to lighten in colour) in the same way as under a white slag. Then a carbide-forming mixture is charged on the slag surface. This mixture consists of coke, lime and fluorspar in the proportion 1 : 3 : 1. The reaction



takes place between the lime and the carbon in the coke at high temperatures in the vicinity of the electric arcs.

Calcium carbide ( $\text{CaC}_2$ ) formed in reaction (28) increases the deoxidising and desulphurising properties of the slag which is known as a carbide slag. The furnace is made air-tight to speed up the formation of calcium carbide. This is done by closing the furnace door and checking the seal of the water-cooled rings in the roof around the electrodes.

A carbide slag contains 55-65%  $\text{CaO}$  and 0.5-0.6%  $\text{FeO}$ ; it has carburising properties and is employed, as mentioned above, in making high-quality steel with a high carbon content ( $>1\%$  C).

*Steel-making procedure without oxidising the charge* (remelting method) differs from the preceding method in that no iron ore is charged into the electric-arc furnace; the heat is conducted with practically no boiling of the bath. Most frequently the furnace is charged with steel waste of a composition near to that of the grade to be made. From 10 to 15 per cent soft iron ( $<0.1\%$  C) is added to the charge to lower the carbon content. It is smelted in an open-hearth furnace and delivered to the electric furnace as an intermediate product.

The primary (ferrous) slag is not removed from the furnace in the remelting method. Thus the critical alloying elements, chromium, titanium and vanadium, are retained and pass from the slag into the metal.

The reducing period in this method of operation may be carried out either under a white or a carbide slag.

**Steel-making induction furnaces.** Induction furnaces that have found application in the steel-making industry in the last 20 or 30 years differ essentially from arc furnaces in the way by which energy is delivered to the molten metal.

The principle of an induction furnace resembles that of a transformer. It has a primary coil (inductor) about which an alternating mag-

netic field is set up with magnetic lines of force of a definite density when an alternating current is passed through the coil. The magnetic flux induces alternating eddy (Foucault) currents in the secondary circuit which comprises a crucible containing the metal charge. The eddy currents heat up and melt the metal.

An induction crucible furnace is illustrated in Fig. 10. The furnace comprises the refractory crucible 1 into which the metal is charged and the inductor 2. The latter is a coil made of copper tubing through

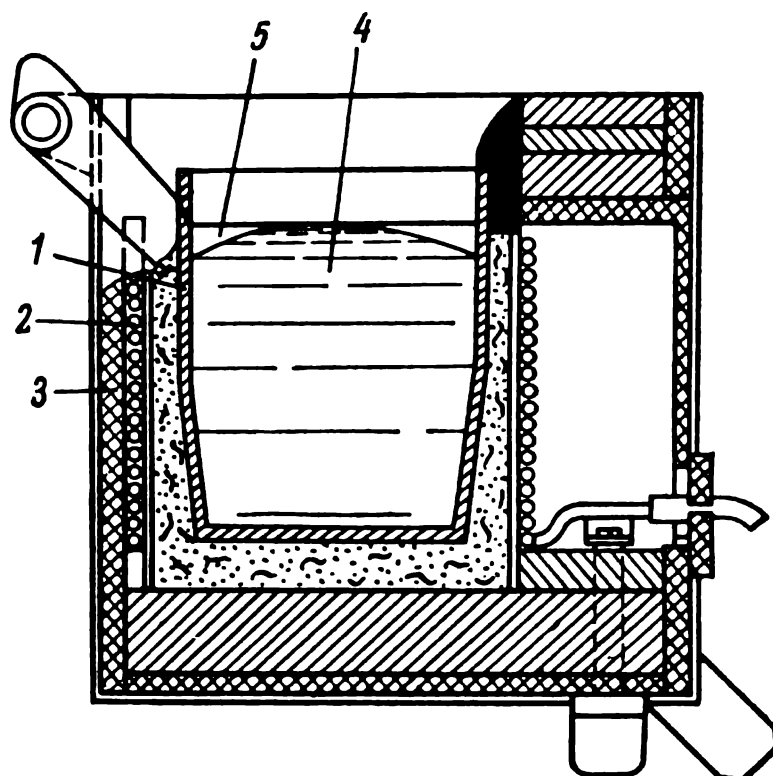


Fig. 10. High-frequency crucible furnace for steelmaking

which cooling water circulates and is arranged within a shell 3 of nonmagnetic material. An insulating lining is provided between the inductor and the crucible. The shell with the crucible can be tilted on horizontal trunnions to pour the molten metal.

The metal to be smelted is charged into the crucible where it is melted down by the heavy secondary currents induced by the magnetic flux of the primary coil. The alternating current for the induction furnace is obtained in special machine or vacuum-tube generators. The leads from the generator to the furnace are either flexible cables or copper busbars. The required power and frequency of the current supplied to the inductor coil depend upon the capacity of the crucible (its diameter) and the type of charge. Induction furnaces usually operate on current with a frequency of 500 to 2500 cps (large-capacity furnaces operate on a lower frequency). The generator rating is selected

so as to obtain from 1 to 0.4 kW per kilogram of the metallic charge.

Crucibles of induction furnaces may have either an acid or a basic lining. An acid lining is made of ground quartzite with a bond of powdered boric acid (up to 1.5 per cent). Well burned magnesite powder ( $\text{MgO}$ ) is used for a basic lining; boric acid (3 per cent) may serve as the bond.

Induction furnaces are available in a wide range of capacities—from 50 kg to 10 tons.

The remelting method of steelmaking is most frequently used in induction furnaces as it entails a negligible loss of alloying elements.

The formation of a layer of slag 5 in the crucible is due mainly to the charging of slag-forming components on the surface of the molten metal 4. The central part of this surface is raised somewhat due to the repulsion of the metal from the stationary turns of the inductor coil upon the passage of an alternating current. The slag temperature in the crucible is less than the metal temperature because the slag has no magnetic permeability and, consequently, no current is induced in it.

An advantage of induction furnaces is that they do not require electrodes; this prevents carburisation of the metal and simplifies control of the process. A vigorous stirring effect (also called the motor effect) is produced by the electromagnetic forces in the crucible. This circulation of the metal accelerates chemical reactions and enables homogeneous metal to be obtained.

Induction furnaces are most often employed in making high-alloy steels and special-purpose alloys. The furnace shell with the crucible may be enclosed in a metal container or tank which can be easily evacuated or filled with a controlled atmosphere of any desired composition (hydrogen, for example). This furnishes extensive possibilities for the application of induction furnaces for vacuum melting and pouring of steels, and for steelmaking in a neutral atmosphere.

### **3-4. Duplex Steel-making Processes**

Combination methods of steel manufacture (duplex processes) are carried out in two steel-making units.

The following combinations are applicable:

- (1) basic and acid open-hearth furnaces;
- (2) a basic open-hearth furnace and a basic electric furnace;
- (3) a Bessemer converter and a basic open-hearth furnace.

In some cases duplex processes prove highly expedient and are the only feasible methods that can be applied.

A combination of a Bessemer converter with a basic open-hearth furnace leads to better utilisation of the converter capacity (each

heat lasts from 20 to 25 minutes); the open-hearth furnace eliminates such disadvantages inherent in the Bessemer process as the inability of removing sulphur, phosphorus and surplus nitrogen. The conversion of pig iron into steel is completed, in the main, within the converter. In this type of duplex process the liquid pig iron is first blown by air in the converter, and then the liquid semifinished steel is charged into the basic open-hearth furnace where the sulphur and phosphorus are removed and the steel is brought to the specified chemical composition.

Combining the operation of a basic open-hearth furnace with that of a basic electric-arc furnace enables the electric energy consumed in melting and desulphurising the metal to be substantially reduced. Here melting down, desulphurisation and, to some extent, boiling are done in the open-hearth furnace. Then the molten steel is poured into the electric furnace where it is subjected to final refining, de-oxidation and alloying.

The expedience of applying duplex steel-making processes in a metallurgical plant depends upon the actual engineering and economic conditions existing at the plant.

### 3-5. Tapping and Pouring the Steel

The finished steel is tapped from the furnace into a well heated steel ladle of the required capacity.

The size of the ladle is usually selected so that it accommodates the whole heat from the furnace and part of the slag which is required as thermal insulation of the metal surface.

From the ladle the steel is poured into metal moulds to produce ingots or into sand moulds to obtain steel castings.

Depending upon the kind of ingots to be produced, the moulds may have a square, round, rectangular or other cross section (fluted moulds). A definite height-to-cross section ratio has been established in practice for ingot moulds; the height should be 5 or 6 times more than the diameter or size of the cross section. The mould walls are tapered to facilitate stripping from the solidified ingots.

Ingots are produced in a wide size range—from 100 kg to 100 tons. Ingots weighing from 6 to 8 tons are the most common size; they are used in rolling departments. Heavier ingots, designed for special purposes, are seldom poured.

In *the top pouring of steel* (Fig. 11) each mould is filled with liquid steel separately.

*Bottom pouring* (Fig. 12) consists in teeming the steel from the ladle into the funnel of a vertical runner (called the fountain) from which the steel enters several moulds (2 to 6 pieces) through narrower runners.

The advantages of top pouring over bottom pouring lie in the greater opportunities for nonmetallic inclusions to become separated from the metal and in that the hotter metal is in the top of the ingot. Bottom pouring provides better surface quality of the ingot and hotter metal at the bottom.

As the liquid metal poured into the mould cools, its volume is reduced thereby leading to shrinkage of the ingot. First to solidify is the outside of the ingot adjacent to the mould walls; the interior part of the ingot remains in a liquid state for some time after the mould is teemed. A shrinkage cavity, or pipe, is formed in the upper central portion of the ingot where the metal continues to be in the molten state for a long time.

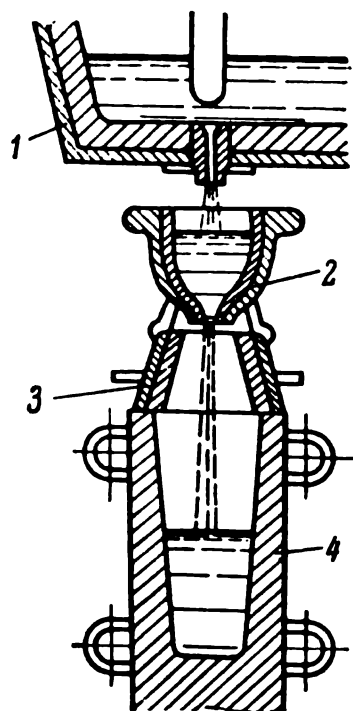


Fig. 11. Top pouring of steel:

1—ladle; 2—tundish;  
3—hot top; 4—mould

The extent of the pipe in the ingot can be reduced somewhat if solidification in the upper part of the mould is retarded. This is done by applying a hot top. Hot tops are extensions of the moulds and are lined with a refractory material. Due to the poor heat conductivity of the hot top walls the metal in this part of the mould is retained in the liquid state for a comparatively long time and a considerable part of the pipe is transferred to the ingot top which is subsequently cut off together with the pipe.

In some cases the metal in the hot top is additionally heated with a gas torch or by introducing a thermit mixture. Such measures reduce the volume of the pipe.

A large economy is achieved in *the continuous casting of steel*, a new method that is being applied at a number of Soviet plants. A diagram showing this method is given in Fig. 13. Liquid metal runs in a continuous stream from the ladle 1 into the intermediate device 2 and then further, into the water-cooled solidifiers 3. A steel seeder is placed in the bottom of each solidifier. This seeder prevents the first portion of the molten metal from dropping straight through the solidifier. Upon contact with the seeders and water-cooled walls the liquid metal begins to solidify rapidly on the ingot surface. It welds to the seeders with which it is pulled downwards, out of the solidifier, by rollers 4 and 5. Solidification of the continuous ingot produced in the solidifier is intensified when it passes through the zone of secondary cooling by water.

The completely solidified ingot next passes to the gas-cutting trucks 6. Continuing to travel downwards together with the trucks

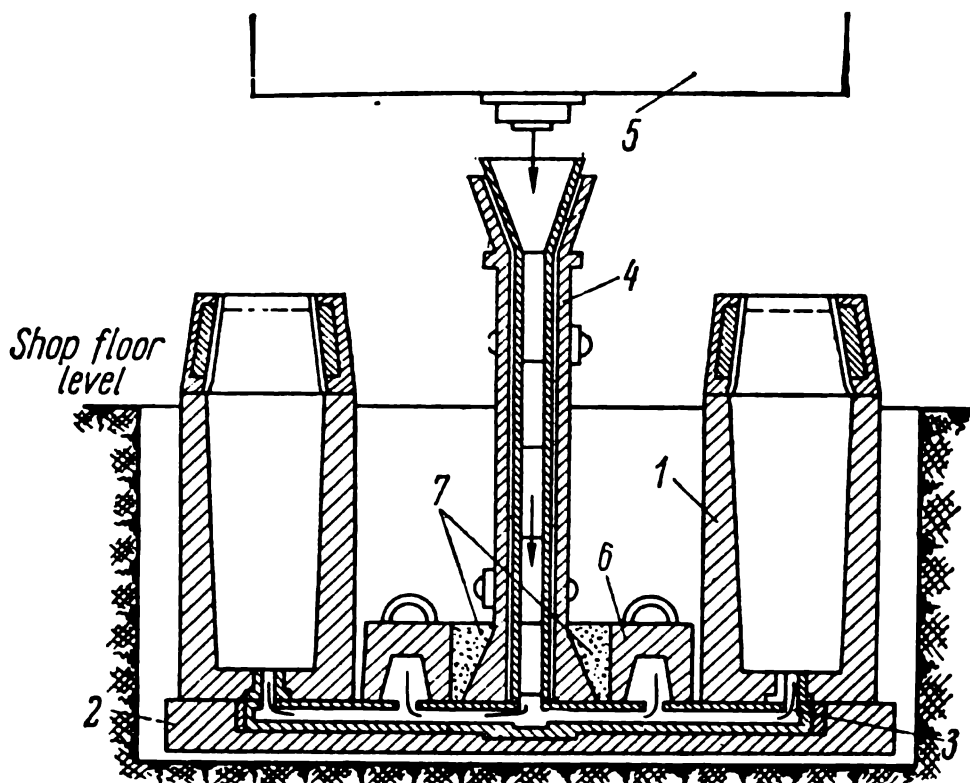


Fig. 12. Mould assembly for bottom pouring of steel:  
 1—mould; 2—mould stool; 3—runners for feeding the metal; 4—central fountain; 5—ladle; 6—weights; 7—fastening components of the fountain

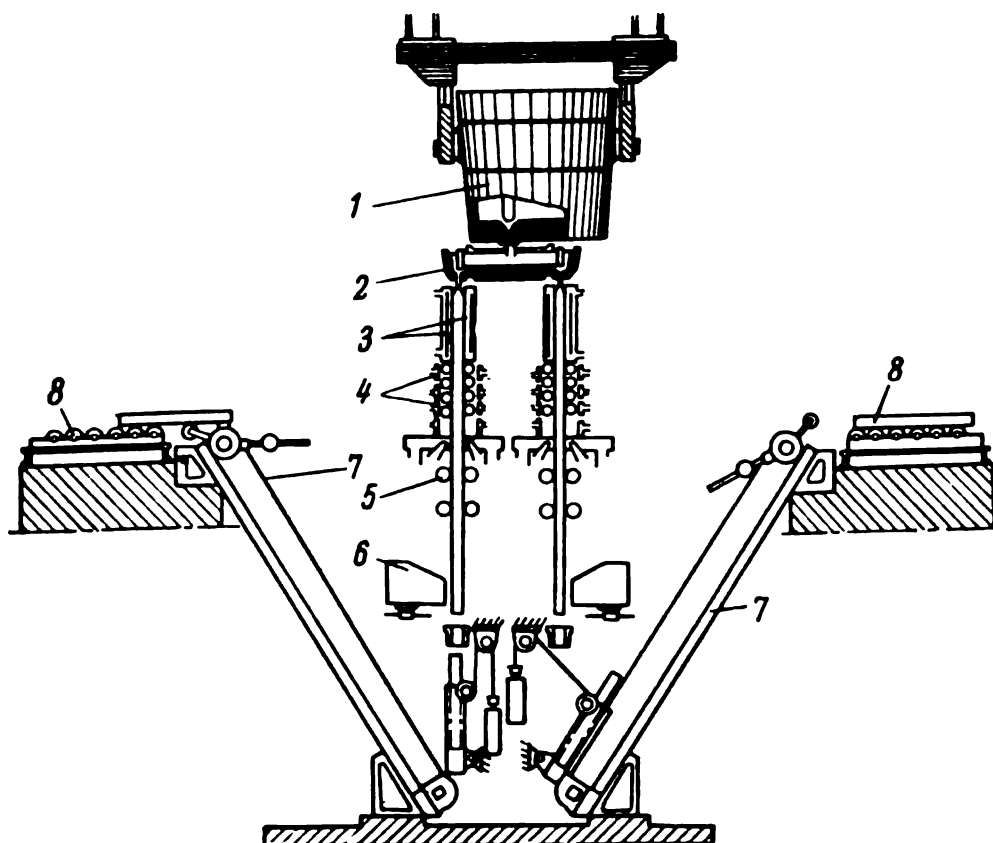


Fig. 13. Principle of the continuous casting machine employed at the Krasnoye Sormovo Plant in the U.S.S.R.



the ingots are cut into billets of definite length which are carried by conveyers 7 over the inspection table 8 and are delivered to the storehouse. The billets are subsequently rolled into sections of the required shape and size.

*Vacuum casting of steel* has proved to be a very effective and efficient method. In this procedure steel, made in any unit, is held in a closed chamber from which air and gases are continuously evacuated. As a result of this treatment the steel has a minimum content of gases and nonmetallic inclusions.

Vacuum treatment is usually carried out in the ladle before pouring the moulds.

This method of producing high-quality metal is not as widely applied as it deserves.

### 3-6. Structure of a Steel Ingot

The liquid metal acquires an inhomogeneous structure after it solidifies in the ingot mould. The reason for this is not difficult to understand if we consider the course of solidification of killed, well deoxidised steel in the mould (Fig. 14). The liquid metal is poured into the mould at a temperature of from 1540°C to 1560°C and immediately comes into contact with the cold bottom and walls of the mould. It is very rapidly cooled at these places, forming a comparatively thin layer of fine, randomly oriented crystals (zones 10 and 5). After cooling on the surface the ingot is slightly retracted from the bottom and sides of the mould due to shrinkage in the inner layers of metal. Air penetrating into the gap between the ingot and mould retards further cooling of the liquid steel inside the ingot so that new large crystals appear which grow in a direction perpendicular to the cooling surfaces of the mould. Here a zone of elongated columnar crystals 7 and 6 is formed; the crystals in zone 8 have a somewhat inclined position relative to the walls. Heat transfer in the central zone 9 proceeds in all directions and therefore the crystals formed here are in disorder, with randomly oriented main axes. A pipe 2 is formed in the upper central part of the ingot where it is enclosed from above by a bridge 1 of solidified metal. Below the pipe are the blowholes 3 and shrinkage porosity 4.

The structure of a steel ingot depends upon the temperature of the metal during pouring, the linear rate of pouring, the temperature of the mould and the thickness of its walls, the chemical composition of the steel, the deoxidising method employed and a number of other factors.

There are always defects in a steel ingot. The chief of these are the pipe and porosity, nonuniform distribution of elements throughout the ingot (segregation), presence of nonmetallic inclusions, blowholes, cavities, cracks and others.

The formation of the pipe and porosity in the upper part of the ingot is explained by the difference in the specific volumes of the liquid and solidified steel (the volume of the metal is reduced in solidifying).

Segregation in an ingot is the result of redistribution of the constituents when the steel passes from the liquid to the solid state. Carbon, sulphur and phosphorus segregate to a greater extent than the other elements in steel. Nonmetallic inclusions and microscopic blowholes are also non-uniformly distributed throughout the ingot.

Nonmetallic inclusions in steel are derived from erosion of the refractory lining of the furnace, spout and ladle; they are also formed in deoxidising steel with manganese, silicon and aluminium. In addition, particles of slag may get into and be retained in the metal. Nonmetallic inclusions may be of diverse chemical composition (iron and manganese sulphides, ferrous oxide, manganous oxide and silicates).

The quality of steel is impaired by nonmetallic inclusions.

Blowholes in a steel ingot are due to the reduction in the solubility of gases when the metal passes from the liquid to the solid state. In the course of steelmaking the molten metal is amply saturated with such gases as hydrogen, nitrogen, oxygen, etc. As the metal solidifies in the mould the surplus gases are evolved but do not have the opportunity to escape from the ingot and are retained in the form of blowholes located either near the surface or inside the ingot.

The hydrogen or carbon monoxide found in the blowholes may be formed as a result of chemical reactions taking place in the molten steel if it is not sufficiently deoxidised. For instance, carbon monoxide may be formed by the reaction of ferrous oxide and carbon dissolved in the metal. This reaction develops intensively throughout the bulk of the metal in the mould when so-called *rimmed steel* is poured.

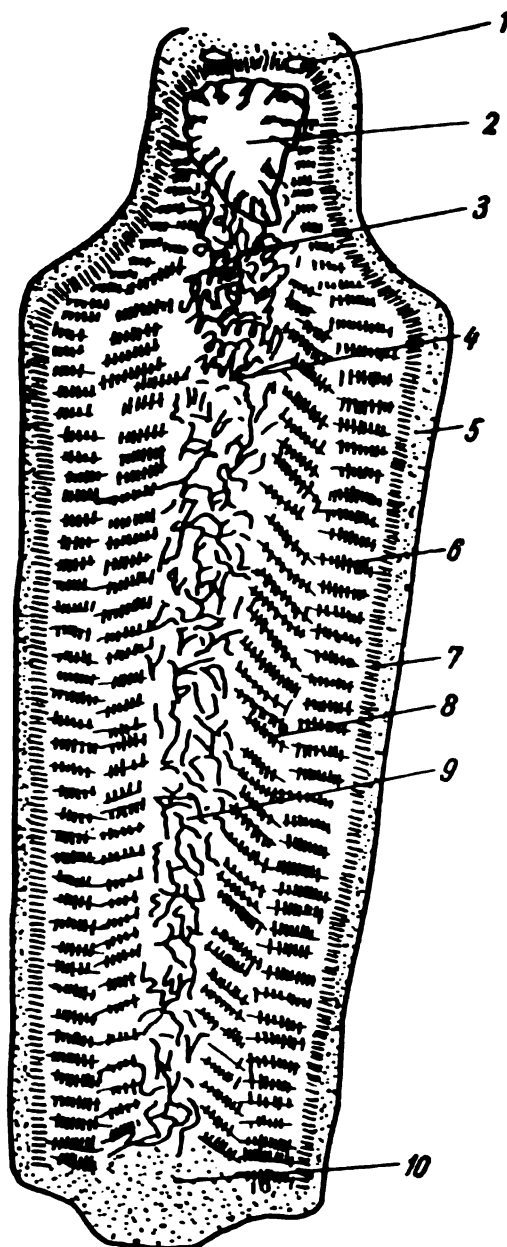


Fig. 14. Structure of a killed steel ingot

Such steel is not deoxidised in the furnace or ladle. The evolved carbon monoxide is collected in the form of blowholes inside the ingot at a depth of at least 15 to 25 mm from the surface. Blowholes in an ingot of rimmed steel are easily and completely welded during subsequent hot rolling. The pipe and segregations of the elements are only slightly evident in ingots of rimmed steel.

Cracks are formed by nonuniform cooling of the various parts of an ingot. Both axial and transverse cracks may be found in ingots. Measures to avoid cracks include careful maintenance of the moulds, strict observance of the prescribed cooling procedure for the steel ingot and others.

The external defects of a steel ingot are removed from the surface by chipping with a chisel in a pneumatic or electric hammer, by grinding with portable grinders or by roughing in special planers, milling machines or other machine tools.

## *Chapter 4*

### THE EXTRACTION OF ALUMINIUM, COPPER AND TITANIUM

#### **4-1. Extraction of Aluminium**

Next to oxygen, aluminium is the most abundant element in nature; about 7.45 per cent of the earth's crust consists of aluminium.

Aluminium is extracted from rock with a high alumina content, the most important sources being bauxite, kaolin, nepheline and alunite.

*Bauxite*, the principal source of aluminium, contains from 30 to 50%  $\text{Al}_2\text{O}_3$ , 3 to 13%  $\text{SiO}_2$ , 2 to 4%  $\text{TiO}_2$ , and 10 to 18 per cent hydrate water. The ore minerals of bauxite are the aluminium hydrates  $\text{Al}_2\text{O}_3 \cdot \text{H}_2\text{O}$  and  $\text{Al}_2\text{O}_3 \cdot 3\text{H}_2\text{O}$ . The less silica in a bauxite the higher its quality as an aluminium ore.

*Kaolin* clays, rich in alumina, contain aluminium in the form of the mineral kaolinite  $\text{Al}_2\text{O}_3 \cdot 2\text{SiO}_2 \cdot 2\text{H}_2\text{O}$ . Kaolin clays are very abundant in nature but the extraction of aluminium from these ores presents difficulties due to the considerable amount of silica present.

The extraction of aluminium comprises three independent processes: (1) converting the aluminium ore into pure alumina, (2) extraction of primary aluminium by the electrolysis of alumina dissolved in molten cryolite, and (3) refining the primary aluminium with chlorine.

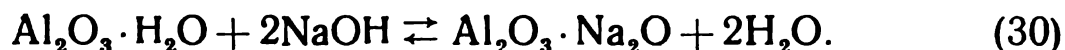
**Extraction of pure alumina.** Methods employed to convert aluminium ores into pure alumina are the alkaline extraction, acid extraction, electric-furnace and combination processes. The choice of any of these methods depends upon the composition and the character of the impurities in the initial aluminium ore available. Aluminium ores, containing as impurities a small quantity of silica but a large amount of iron oxides (high-grade bauxites), are most expediently treated by alkalines since the latter dissolve the silica but not the iron oxides which can, therefore, be easily separated and run to waste. Aluminium ores with a high silica content and with but a small amount of iron oxides (kaolin clays) should be treated by acids since acids dissolve the iron oxides but not the silica. Aluminium ores containing large amounts of both silica and iron oxides can best be treated by either the electric-furnace or combination process in which ferrosilicon is first smelted from the ore and the alumina is transferred to the slag.

In the alkaline extraction process alumina can be obtained either by the wet alkaline method or by sintering. Both methods are based on the conversion of the aluminium ore into a chemical compound soluble in water—sodium aluminate  $\text{Na}_2\text{O} \cdot \text{Al}_2\text{O}_3$ . At  $45^\circ\text{--}50^\circ\text{C}$  this compound is decomposed by hydrolysis so that crystalline aluminium hydroxide is precipitated from the solution. The aluminium hydroxide residue is carefully washed, dried and then roasted to drive off the hydration water:



A new product—pure, dry alumina is obtained as a result of reaction (29).

In the *wet alkaline (autoclave) process* the bauxite is first dried (at  $200^\circ\text{C}$ ) and then crushed and ground in ball mills. The ground bauxite is put into autoclaves (closed vessels) with concentrated alkali and held for 3 or 4 hours at a temperature of  $150^\circ\text{C}$  and a pressure of 5 to 7 atmospheres. The following reaction takes place in the autoclave:



The sodium aluminate ( $\text{Al}_2\text{O}_3 \cdot \text{Na}_2\text{O}$ ) formed here passes into the aqueous solution while the ferric oxide ( $\text{Fe}_2\text{O}_3$ ), titanium dioxide ( $\text{TiO}_2$ ) and other impurities of the ore, not soluble in the alkalis, are precipitated as red mud and removed by filtering. A part of the silica ( $\text{SiO}_2$ ) also passes into the red mud and forms an insoluble compound—sodium aluminium silicate while another part of the silica is dissolved in the alkali



and passes into the aqueous solution, polluting it with silicon.

After being separated from the red mud the strong alkaline solution with sodium aluminate is fed to large vats where it is diluted with water and then undergoes hydrolysis at 40° C. For this purpose the solution in the vat is seeded with freshly precipitated crystalline aluminium hydroxide  $\text{Al}(\text{OH})_3$ , which acts as a catalyser of the reaction



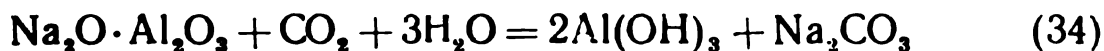
$\text{Al}(\text{OH})_3$  is filtered out, washed and roasted to  $\text{Al}_2\text{O}_3$  according to reaction (29).

In the *dry alkaline (sintering) process* the finely ground bauxite is mixed with commercial soda and the mixture is sintered at 800°-1000° C. Under these conditions sodium aluminate is obtained in the solid form according to the reaction



and is leached with water to form a solution.

When the charge is sintered the undesirable compounds  $\text{Fe}_2\text{O}_3 \cdot \text{Na}_2\text{O}$  and  $\text{TiO}_2 \cdot \text{Na}_2\text{O}$  are formed in addition to  $\text{Al}_2\text{O}_3 \cdot \text{Na}_2\text{O}$ . During leaching of the sintered products with hot water (at 90°-95° C) these compounds are precipitated as a red mud and easily separated from the clear aqueous solution of sodium aluminate  $\text{Al}_2\text{O}_3 \cdot \text{Na}_2\text{O}$ . The clear aqueous solution of sodium aluminate is treated with a stream of carbon dioxide. This causes the reaction



as a result of which  $\text{Al}(\text{OH})_3$  is precipitated. This precipitate is washed with water, dried and then roasted to  $\text{Al}_2\text{O}_3$ .

The *acid extraction process* is begun by dissolving the aluminium ore in sulphuric or hydrochloric acid. The aluminium salts formed undergo further chemical treatment as a result of which  $\text{Al}(\text{OH})_3$  is precipitated from the solutions. This aluminium hydroxide is then roasted to obtain alumina.

**The electrolysis of alumina.** Pure alumina has a high melting point and is a very stable compound of aluminium and oxygen. This compound cannot be decomposed (reduced) to the metal by carbon, hydrogen or carbon monoxide even at the highest temperatures.

To obtain metallic aluminium it is necessary to resort to electrolysis of the alumina in a liquid electrolyte—molten cryolite. Cryolite is a double fluoride of sodium and aluminium  $\text{Na}_3\text{AlF}_6$  which is produced from fluorspar ( $\text{CaF}_2$ ), sulphuric acid, aluminium hydroxide and soda. Cryolite has a lower melting point than alumina and is capable of dissolving alumina.

Alumina undergoes electrolysis in electrolysing baths, also called aluminium reduction cells (Fig. 15). The bottom 1 and the walls 2

of the cell are made of carbon blocks. Thick copper busbars 7 pass through the bottom and are connected to the negative pole of the power source. Carbon electrodes 4, serving as the anodes and connected to the positive pole of the power source, are suspended above the cell from the copper busbar 3.

Before beginning electrolysis the cell bottom is covered with a thin layer of ground coke. Then the carbon electrodes are lowered into contact with this layer and current is passed between the cell bottom and the electrodes. When the carbon lining of the cell becomes

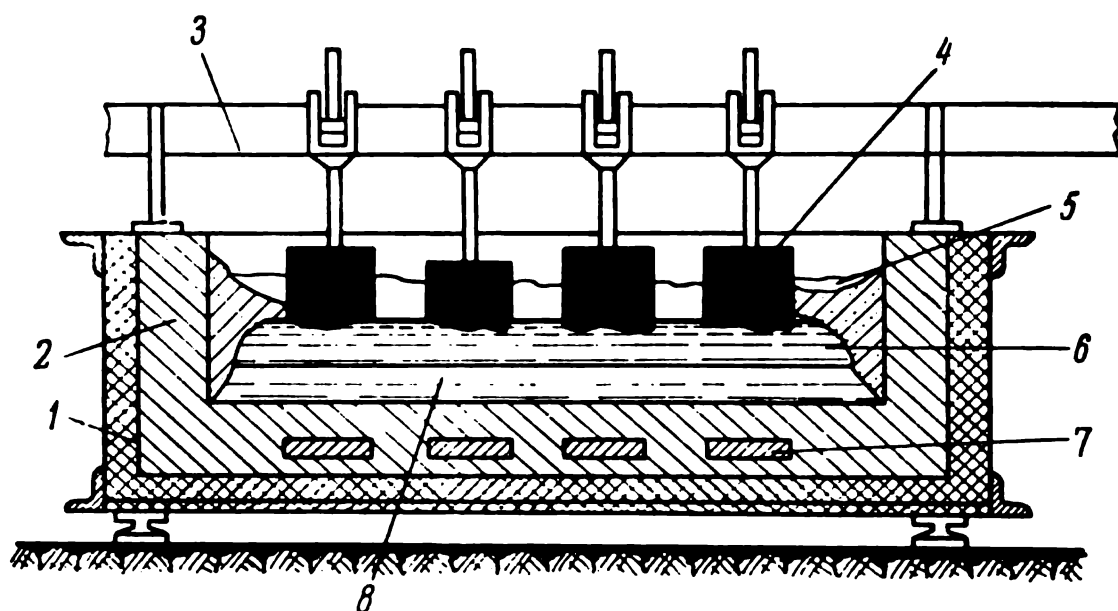
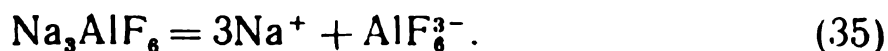


Fig. 15. Principle of the aluminium reduction cell

red hot, cryolite is added gradually and melted, slowly raising the electrodes with an increase in the bath volume. When the layer of molten cryolite 6 reaches a depth of 200-250 mm the alumina 5 in an amount up to 10-15 per cent of the cryolite is charged into the bath where it dissolves.

Under the action of the direct current the molten cryolite partly dissociates into ions according to the reaction



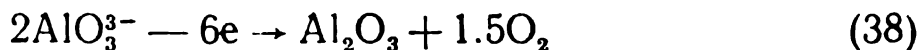
The alumina dissolved in the cryolite also dissociates according to the reaction



The positively charged ions  $\text{Al}^{3+}$  formed as a result of reaction (36) are carried over by the electric current to the carbon bottom, i.e., the cathode of the cell, where a cathodic process takes place which involves aluminium ion discharge and the liberation of metallic aluminium



The negatively charged anions  $\text{AlO}_3^{3-}$  are carried by the current to the carbon electrode, the anode, where the anodic process



results in the formation of gaseous oxygen and alumina.

At the electrolysis temperature of  $930^\circ\text{--}1000^\circ\text{C}$  the oxygen formed on the electrodes reacts with their carbon which becomes partly oxidised.

The positively charged ions of sodium  $\text{Na}^+$  and the negatively charged ions  $\text{AlF}_6^{3-}$  formed as a result of reaction (35) also carry over the electric current but they do not participate in the primary discharges because they have other deposition potentials on the cathode and anode than  $\text{Al}^{3+}$  and  $\text{AlO}_3^{3-}$ .

The molten aluminium 8 gradually collects on the bottom of the bath under the layer of cryolite 6 from where it is removed by siphoning.

The alumina content of the cryolite solution decreases in the course of electrolysis; when it becomes as low as 1 or 2 per cent small electric arcs appear on the surface of the carbon electrode at the places where it contacts the electrolyte and the voltage rises sharply. This phenomenon is called the anode effect. This effect can be eliminated by immediately charging a new portion of alumina, and normal operation of the cell will be restored.

From 80 to 100 aluminium reduction cells of this type are connected together in series; they operate at 4 to 4.3 V per cell and with a current of 40,000 to 100,000 A.

The production of one ton of aluminium requires about 1.98 tons of alumina, 0.1 ton of cryolite, 0.6 ton of carbon electrodes (anodes) and from 16,600 to 18,000 kWh electric energy.

As a rule, the liquid aluminium is tapped once every 3 or 4 days. A bath can run continuously for 2 or 3 years.

**Refining primary aluminium.** The primary aluminium obtained in the reduction cell contains a number of impurities and must therefore be refined to remove them.

Aluminium is most frequently refined in closed ladles with a capacity of from 1200 to 1300 kg by blowing chlorine through it at a temperature of  $750^\circ\text{--}770^\circ\text{C}$  for 10 or 15 minutes, after which the metal is cast into ingots.

After remelting in electric furnaces a certain amount of primary aluminium is cast into plate billets for subsequent rolling into sheets.

Sometimes electrolytic refining is employed to eliminate the impurities dissolved in the aluminium. Here the impure aluminium serves as the anode and undergoes dissolution while the pure aluminium is the cathode. An electrolyte layer of anhydrous chlorous and

fluoric salts is provided between the impure and pure aluminium. Electrolytic refining produces aluminium with a purity of 99.9 to 99.8 per cent.

#### 4-2. Extraction of Copper

The raw materials for the production of copper are sulphide or oxide copper ores. Most of the copper produced is smelted from sulphide ores (about 80 per cent) while oxide ores account for only 15 to 20 per cent. Sulphide ores are more widespread in nature due to the higher affinity of copper for sulphur than for oxygen.

The most abundant copper sulphide ore is copper pyrite containing the mineral chalcopyrite ( $\text{Cu}_2\text{FeS}_4$ ). In some cases, the so-called copper glance is used; it contains the mineral chalcocite ( $\text{Cu}_2\text{S}$ ). Of the oxide ores one containing the mineral cuprite ( $\text{Cu}_2\text{O}$ ) deserves mention.

**Concentration of copper ores.** All copper ores are very lean as they contain only from 1 to 5% Cu. Therefore, before smelting they must be concentrated by flotation.

In flotation the copper ore is first crushed finely and then ground in ball mills to grains from 0.05 to 0.5 mm in size. A certain quantity of flotation reagents (100 to 300 g per ton) is added to the milled ore. These promote the formation of a film on the surface of the grains that is poorly wettable with water and that reduces the surface tension of the water.

The ore treated in this manner is charged into the flotation machine where it is mixed with water to form a pulp. This pulp is treated with dispersed streams of air. The fine grains of ore minerals with low wettability by water adhere to the air bubbles and rise with them to the surface forming a foam. The foam is continuously removed from the machine and, after filtration and drying, provides the copper flotation concentrate. Particles of rock, having a high wettability, sink to the bottom of the machine and are run into waste as tails.

Flotation enables lean copper sulphide ores to be converted into a concentrate containing from 15 to 20% Cu.

Before smelting, the copper concentrate and rich copper sulphide ores are subjected to an oxidising roasting process at  $600^\circ\text{--}900^\circ\text{C}$  in which part of the sulphur is removed in the form of a gas (this gas is trapped and utilised in the production of sulphuric acid).

**Smelting to obtain blister copper.** Following the oxidising roasting, the copper sulphide ore and the concentrate are conveyed to either a blast or reverberatory furnace where they are smelted into matte. Matte is a melt consisting mainly of two sulphides— $\text{Cu}_2\text{S}$  and  $\text{FeS}$ —and a small amount of impurities. Most often the matte contains 30 to 50% Cu, 40 to 20% Fe and 25 to 22% S; it has a melting point of  $950^\circ\text{--}1000^\circ\text{C}$ .



The molten matte serves as the initial material in making blister copper. This is done by pouring the matte into a horizontal cylindrical converter and forcing thin streams of air through the liquid mass. Fig. 16a illustrates a modern copper converter while Fig. 16b is its cross section.

The steel shell 1 of the converter is cylindrical in shape; its flat end walls 2 are reinforced by beams. Two heavy rims 4 encircle the converter shell; each rests on two pairs of rollers 3 on which the converter can be rotated. The rollers are mounted in support brackets 5 of the foundation 6.

The converter is rotated by the electric drive 8 which is linked through gearing to one of the shell rims. A mouth 7 serves to charge the molten matte, to skim the slag and to pour out the blister copper. Ground quartzite is charged by air into the converter in the course of operation through funnel 12 inserted into a hole in an end wall. Compressed air from the blast pipeline 9 enters the tuyeres 10 which are arranged in the side of the converter. The lining 11 is made of magnesite brick.

There are two main stages in converting copper mattes. In the first (initial) stage the oxygen of the blast almost completely oxidises the iron in the matte, and the copper to some extent as well. The iron oxidation reaction is



The ferrous oxide formed in reaction (39) rises to the surface where it reacts with the quartz, previously introduced into the converter, to form a slag:



Reactions (39) and (40) evolve a great deal of heat and are the source of heating the melt during the blow.

The ferrous silicate slag produced in the first stage is poured off and the second (final) stage begins. During the blow of the second stage the cuprous sulphide is oxidised by the reaction



which is also accompanied by an evolution of heat. Cuprous oxide formed in reaction (41) dissolves in the melt and reacts with the cuprous sulphide:



thereby producing metallic copper. In contradistinction to reactions (39), (40) and (41), reaction (42) proceeds with the absorption of heat and requires that the bath be well heated beforehand.

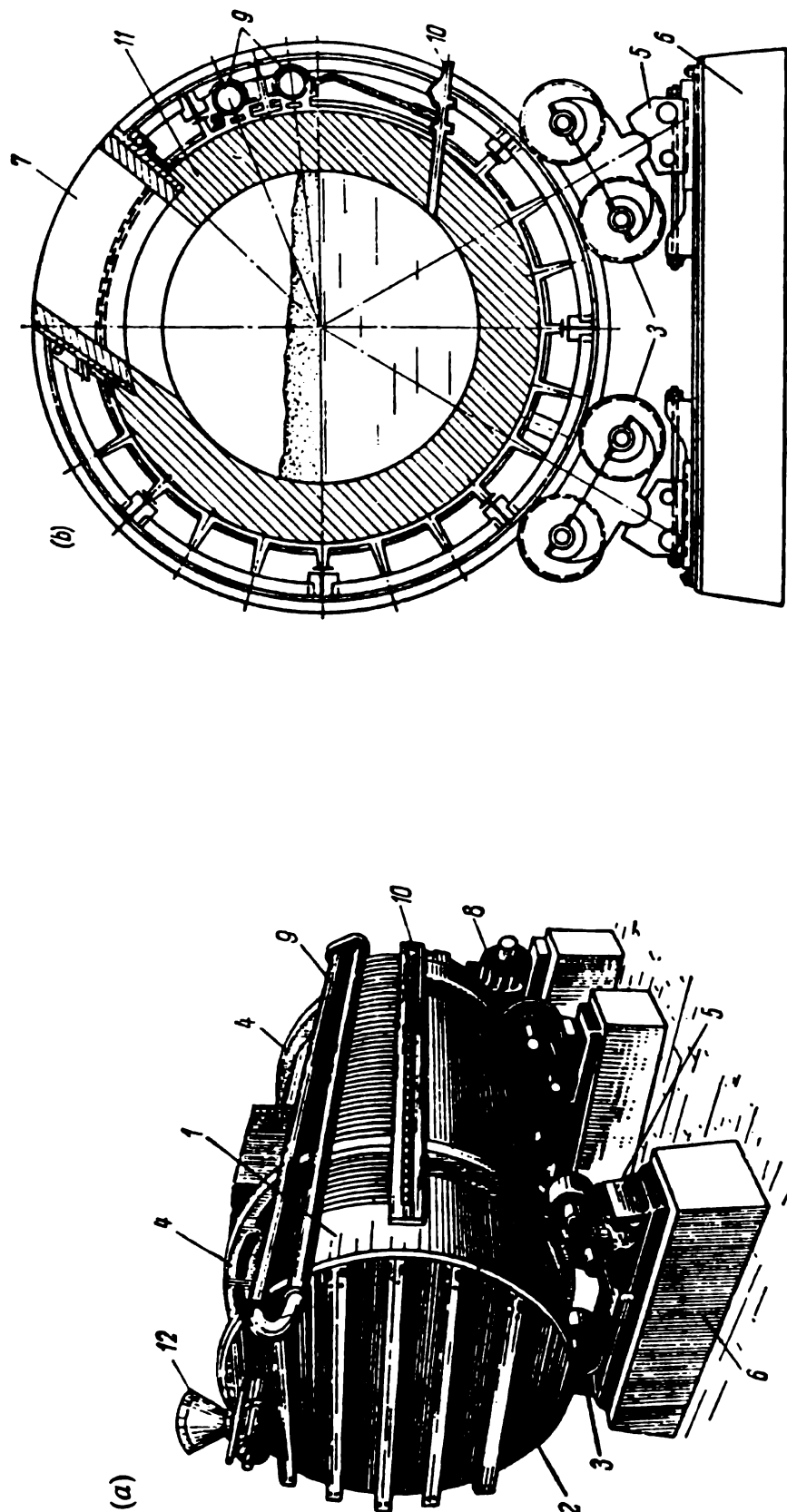


Fig. 16. Copper converter  
(a) external view; (b) cross section

The blister copper produced in the converter is cast in flat ingot moulds on a straight-line casting machine. The average composition of blister copper is: 98.5 to 99.5% Cu, 0.3 to 0.5% S, 0.03 to 0.1% Fe, up to 1% O, 0.3 to 0.5% Ni and traces of As, Pb, Sb and the noble metals.

**Copper refining.** The impurities of blister copper are eliminated by fire or furnace refining followed by an electrolytic process.

In *fire refining* the blister copper is charged into a flame furnace with a capacity up to 250 tons where it is melted in an oxidising atmosphere. Under these conditions only those impurities are removed and go into the slag that have a greater affinity for oxygen than copper has.

The refining process is speeded up by blowing air at a pressure of 1.5 to 2 atm through iron tubes into the molten copper bath. The order in which the impurities (aluminium, silicon, manganese, tin, iron, arsenic, antimony and lead) are oxidised depends upon the temperature of the process and the concentration of these impurities in the liquid metal. Most of the impurities go into the slag as oxides ( $\text{Fe}_2\text{O}_3$ ,  $\text{Al}_2\text{O}_3$ ,  $\text{SiO}_2$ , etc.) but some are removed with the furnace gases in the form of vapour. All noble metals remain completely in the copper after fire refining.

The copper itself is, of course, oxidised in the blow and absorbs a sufficiently large amount of oxygen. Therefore, before tapping the liquid metal it is deoxidised.

After deoxidation the copper becomes more dense and it is cast into ingots or into anodes for subsequent electrolytic refining.

*Electrolytic refining* is practised to obtain copper of higher purity and also to extract the noble metals. Electrolytic refining is performed in wooden tanks lined with lead or other protective materials. The tank is filled with the electrolyte consisting of an aqueous solution of sulphuric acid (10 to 16 per cent) and a definite quantity of copper sulphate. Then the copper anodes to be refined and cathodes in the form of pure electrolytic copper starting sheets from 0.5 to 0.7 mm thick are lowered into the electrolyte. The anodes are connected to the positive pole of the power source and the cathodes to the negative pole. Copper busbars for conducting the current are arranged at the sides of the tank. Electrolysis takes place when direct current passes through the electrolyte. The anodes are dissolved in the bath and then the positively charged ions of copper discharge on the cathode where pure copper is deposited. The noble metals in the copper sink to the bottom of the tank and pass into the slime.

Pure copper is deposited on the cathode during a period of 10 to 12 days. Each cathode then weighs from 60 to 90 kg. With a current density of 100 to 300 A per sq m, a current of 10,000 A and a voltage of 100 to 200 V, from 250 to 350 kWh are required for electrolytic refining of one ton of copper.

### 4-3. Extraction of Titanium

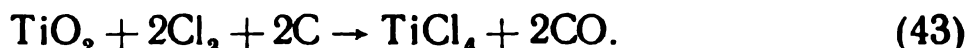
Titanium is one of the most abundant metals; its content in the earth's crust is next only to aluminium, iron and magnesium.

Of the numerous titanium minerals known at the present time ilmenite  $\text{FeTiO}_3$  is of industrial interest. It contains 52.65 per cent titanium dioxide ( $\text{TiO}_2$ ).

There are three allotropic forms of titanium dioxide: rutile, anatase and brookite. Of these rutile is the most abundantly available in nature.

Ilmenite is found in placer deposits in mixtures with magnetite ( $\text{Fe}_3\text{O}_4$ ) and hematite ( $\text{Fe}_2\text{O}_3$ ); it forms ores known as titanomagnetites containing up to 20 per cent titanium.

Though a great many methods have been devised for reducing titanium dioxide, not one of them has proved satisfactory for production on a commercial scale. At the present time much metallic titanium is obtained by the reduction of titanium tetrachloride ( $\text{TiCl}_4$ ) which is produced in the reaction

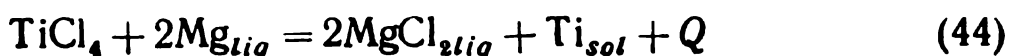


This process consists in mixing rutile with crushed coke and calcinating at a temperature of  $800^\circ\text{C}$ . In chlorination the titanium tetrachloride is evolved as a vapour together with the impurities which are collected in the condenser of the plant. The titanium tetrachloride condensed from the gases has the following percentage composition:

$\text{TiCl}_4$	$\text{SiCl}_4$	$\text{FeCl}_3$	$\text{Cl}_2$	Others
97 to 99	1.5 to 2.5	0.02 to 0.002	0.1 to 0.3	up to 0.1

The nonvolatile matter ( $\text{FeCl}_2$ ,  $\text{MgCl}_2$ ,  $\text{CaCl}_2$ ,  $\text{NaCl}$ , etc.) is precipitated to the bottom of the plant and is separated by filtration from the titanium tetrachloride and other impurities passing into solution during chlorination. Then commercially pure titanium tetrachloride is separated from the impurities by rectification, making use of the different boiling points of the chlorides ( $\text{TiCl}_4$ — $136^\circ\text{C}$ ,  $\text{FeCl}_3$ — $319^\circ\text{C}$  and  $\text{SiCl}_4$ — $57^\circ\text{C}$ ).

The next step is to reduce the titanium tetrachloride. This may be done with magnesium at temperatures of  $800^\circ\text{--}900^\circ\text{C}$ . A predetermined quantity of ingot magnesium is charged into a special crucible from which the air is evacuated and to which an atmosphere of pure argon is introduced. Liquid  $\text{TiCl}_4$ , fed into the reaction chamber, vaporises and reacts with the molten magnesium to form solid particles of reduced metallic titanium. The particles of metallic titanium sinter to form a sponge:



The liquid magnesium is removed. The sponge contains 99 per cent pure titanium. Ductile metal can be produced from the sponge either by a metallurgical procedure (melting) or by a powder metallurgy technique (compacting and sintering). The more efficient method is electric arc melting of the sponge in an argon atmosphere in a water-cooled crucible. The metal obtained in this way contains 99.8% Ti.

Other methods for the reduction of titanium are also used. Highly pure titanium (99.95% Ti) is produced by the iodide method which consists in decomposing halides of titanium (usually iodides) on an incandescent titanium or tungsten filament. The chemical reaction  $\text{Ti} + 2\text{I}_2 \rightleftharpoons \text{TiI}_4$  proceeds to the right at temperatures from 500° to 550°C while at temperatures from 1300° to 1400°C it proceeds to the left. In practice the titanium sponge is charged into the annular space between the wall of the reaction vessel and a molybdenum cylinder. The reaction vessel is heated to 500°-550°C and the titanium filament is heated by electric resistance to 1300°-1400°C. Vapours of  $\text{TiI}_4$  reach the filament and decompose so that titanium is concentrically deposited on the filament.

**Melting and Casting of Titanium and Its Alloys.** Two types of electric arc furnaces are employed to melt titanium. In the first type the ingot is fused on in a cylindrical crucible with a fixed bottom. The crucible is made of copper and is cooled by water. The level of the metal bath rises as the material is charged into the furnace. The furnace is furnished with an electrode raising mechanism which enables a constant arc length to be maintained. In the second type of furnace the electrode is fixed at a definite height. A constant bath level is maintained in charging by continuously pulling the ingot downwards. The ingot is clamped to a descending holding plate. Melting is carried out in an inert gas (argon) atmosphere.

Titanium melted in a furnace with a nonconsumable electrode becomes polluted with the electrode material. Therefore, in industrial arc furnaces it proves more expedient to employ graphite electrodes coated with a layer of titanium carbide instead of tungsten electrodes. The pollution of the titanium may be avoided by using consumable electrodes made of titanium or its alloys. Ingots of titanium up to 2 tons in weight are obtained by means of this arc melting method. Another method consists in melting titanium in a graphite crucible. The metal in the crucible is heated either by high-frequency induction or by the radiation of a graphite heater. Melting is conducted in a vacuum or in an inert gas atmosphere. Titanium is carburised to a degree of 0.7-0.8 per cent.

Titanium alloys can be produced by a drip method. The material to be remelted, in the form of a bar, is hung up in a crucible surrounded by a high-frequency induction coil. The lower end of the

bar melts and runs down into a copper ingot mould. Melting in a suspended state in an electric field is also sometimes used. In this method the high-frequency current both heats the metal and supports it in space by means of a magnetic field built up around two induction coils arranged above and below the specimen being melted.

The moulding materials used in making castings of titanium and its alloys must be stable in respect to molten titanium and possess a high degree of degasification. Permanent moulds are made of graphite but the carburisation of the titanium cast in them may reach 1 per cent.

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## PRINCIPAL PROPERTIES OF METALS AND ALLOYS

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### Chapter 5

#### STRUCTURE AND CONSTITUTION OF METALS AND ALLOYS

##### 5-1. Crystalline Structure of Metals

All solids are either amorphous or crystalline.

*Amorphous solids* (e.g., glass, tar, resin and others) are characterised by a completely random (chaotic) arrangement of their constituent particles. A definite geometrical order, on the contrary, is observed in the internal arrangement of the constituent particles, atoms or ions, of *crystalline solids* which include all metals. These particles are located in the solid in such a manner that in the aggregate they form the space or crystal lattice.

The space lattice of any solid is comprised of innumerable conjugate unit cells inside of which the atoms (or ions) are arranged in a definite order. Fig. 17 illustrates the arrangement of atoms (or ions) in the unit cells of three types of crystal lattices: body-centred cubic, face-centred cubic and close-packed hexagonal.

The unit cell of a body-centred cubic space lattice is a centred cube with nine atoms of which eight are located at the corners of the cube and the ninth at the centre.

The unit cell of a face-centred cubic space lattice has fourteen atoms of which eight are located at the corners of the cube and six at the centres of the six faces. This lattice has a more compact packing of the atoms than the preceding type.

A still more compact arrangement of the atoms is observed in a close-packed hexagonal space lattice.

The crystal lattices of various substances differ not only in the arrangement of their atoms but also in their linear dimensions (lattice constants or parameters) which represent the distances between atoms in adjacent parallel planes of the unit cell. Thus, for example,

the lattice constant for a cubic space lattice is the length of the edge  $a$  on the unit cell (see Fig. 17a). In the hexagonal lattice the unit cell has two constants— $a$  and  $c$ .

The distance between neighbouring atoms (or ions) at the lattice points is very small and is measured in Angström units ( $\text{\AA}$ ) equal to  $10^{-8}$  cm or in kilo-X units ( $1\text{kXU}=1.00202\times 10^{-8}$  cm).

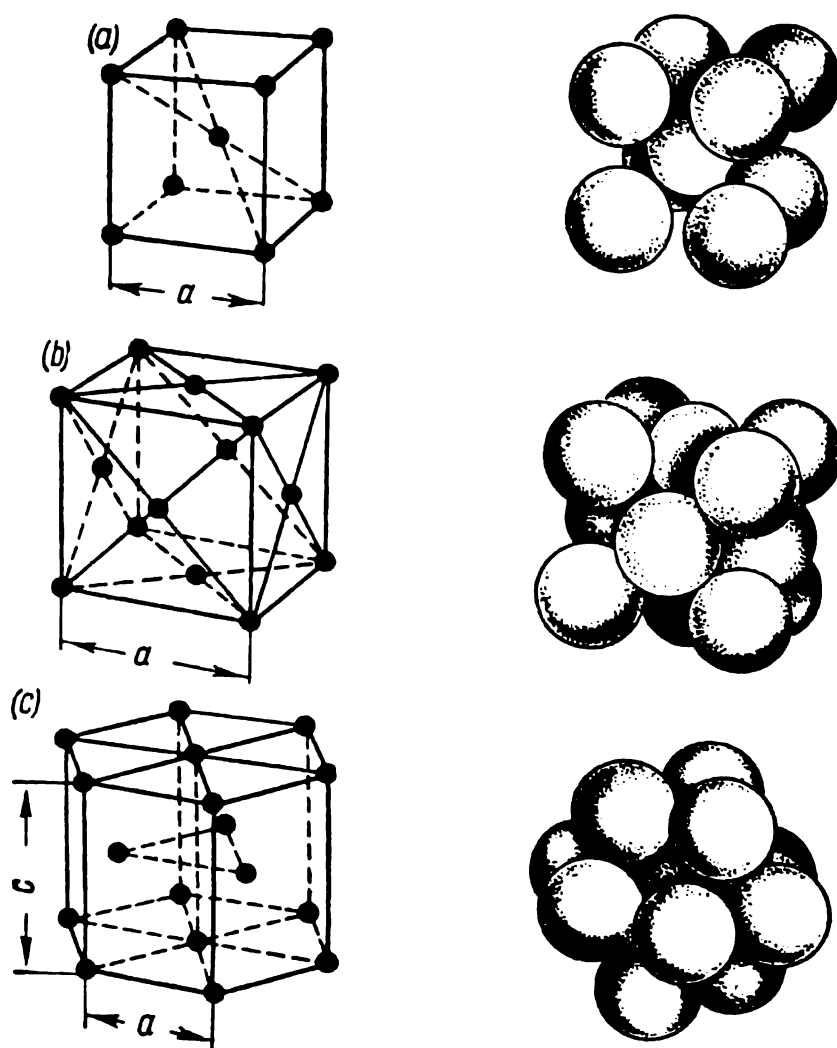


Fig. 17. Atomic arrangement in unit cells of crystal lattices:

(a) body-centred cubic; (b) face-centred cubic; (c) close-packed hexagonal

Another essentially important characteristic of a crystal lattice is the number of equally-spaced nearest neighbours that each atom has. This is called the coordination number and determines the closeness with which the atoms are packed in the given type of crystal lattice. Each atom of a simple cubic unit cell, for instance, has six nearest neighbours. The coordination number of this cell is equal to 6 and is designated as C6. In the body-centred cubic lattice each atom has 8 neighbours and its coordination number is 8 (C8).



Experience shows that mechanical properties of crystalline solids may differ greatly if measured in different directions on the specimen. This feature of solids is called *anisotropy* and is easily explained by their atomic structure. The space lattice of a solid is built up by innumerable repetition of unit cells containing properly arranged atoms. If the space lattice is cut by planes in various directions the number of atoms and the minimum distance between them may vary for each plane. This has an effect on the bond (attraction) between the atoms and, therefore, on the mechanical properties of crystalline solids measured in various directions.

The mechanical properties of amorphous solids are practically the same in all directions; this feature is called *isotropy*.

Many metals (such as iron, tin, zinc, nickel, cobalt, etc.) are subject to crystalline transformations in the solid state upon changes in their temperature. The existence of the same metal in two or more crystalline forms, differing in the arrangement of the atoms of the unit cell in the space lattice, is called *allotropy*. The process in which the crystal lattice is changed in accordance with the temperature is called an *allotropic* or *polymorphic transformation* of the metal. Allotropic transformations are accompanied by either the evolution or absorption of heat.

The allotropic forms in which a metal exists are called its *modifications*. The different modifications of the same metal are designated by the first letters of the Greek alphabet:  $\alpha$ ,  $\beta$ ,  $\gamma$ ,  $\delta$ , etc.

A vital feature of metals is their plasticity, i.e., their ability to change their form (be deformed) under the action of external forces without the violation of integrity.

Metals also have high thermal and electrical conductivity and a characteristic metallic lustre.

Atoms at the points of a crystal lattice continuously oscillate in what is called thermal vibration. The higher the temperature of the metal, the larger the amplitude of their oscillation. When the metal is heated to the melting point, the amplitude of this oscillation reaches a certain critical value at which the crystal lattice is destroyed and the metal passes from the solid to the liquid state.

**Freezing (solidification) of metals.** When the temperature of liquid metal drops below a certain critical value the metal begins to solidify. As a result the metal is completely transformed into the crystalline state. The cooling curve shown in Fig. 18 illustrates the transformation of metal from the liquid to the solid state under slow cooling conditions (low rate of cooling). The first centres of crystallisation or nuclei appear, randomly distributed in the volume of the molten metal, at point *m* (temperature  $t^\circ$ ). New nuclei form within the solidifying metal as it continues to cool and the previously formed crystals grow in size. The temperature of the metal remains

constant until all of it has solidified (point  $n$ ) since the freezing process is accompanied by evolution of the latent heat of fusion. Thus the horizontal line is obtained on the cooling curve (see Fig. 18)

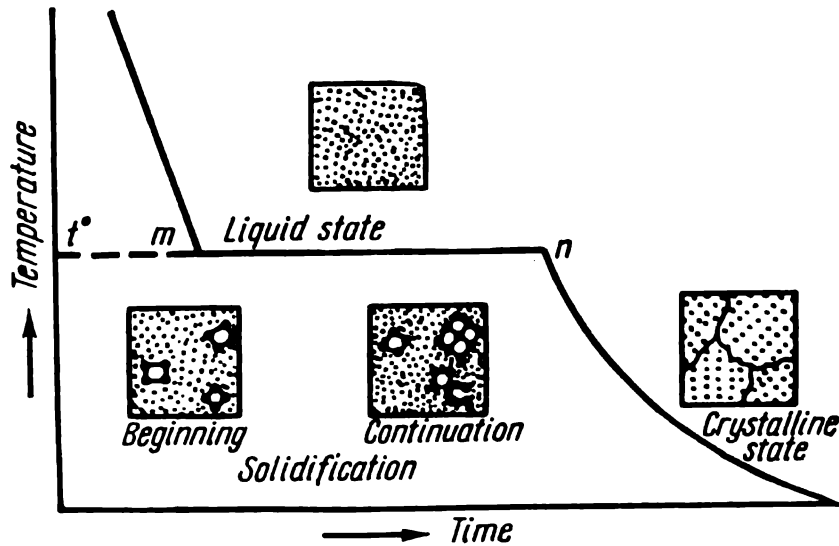


Fig. 18. Freezing of a pure metal (shown schematically)

indicating that the freezing of metal from the liquid state proceeds at a strictly constant temperature. When all the metal has solidified a further drop in temperature is observed.

The constant temperatures at which metals pass from the liquid to the solid state and vice versa are called the *freezing point*, or

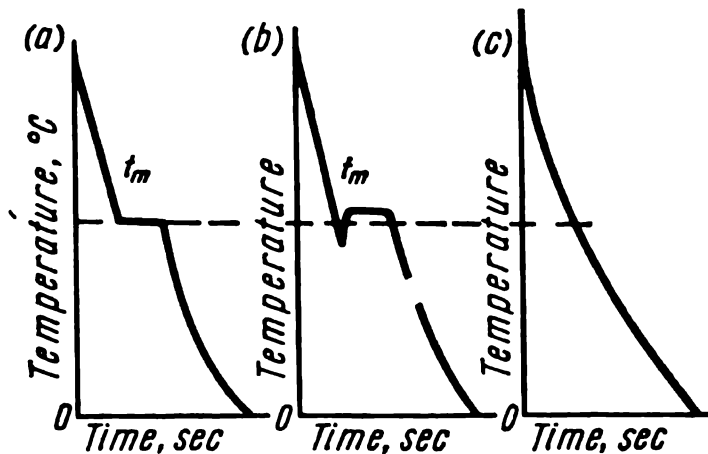


Fig. 19. Cooling curves:

(a) a slowly cooled crystalline solid; (b) a supercooled crystalline solid; (c) an amorphous solid

*freezing temperature*,  $t_f$ , and the *melting point*, or *melting temperature*,  $t_m$ , respectively (Fig. 19a).

At a sufficiently high rate of cooling the liquid metal may be supercooled. This means that the metal will continue to be in the liquid state at a temperature below its usual (equilibrium) freezing point.

The phenomenon of supercooling in the liquid metal can be explained as follows. As a result of the rapid cooling, a large number of nuclei are formed in the liquid metal and the crystals that form grow intensively. This leads to the evolution of a great deal of heat which is capable of maintaining the metal in a liquid state for a certain length of time and even of raising the temperature to the usual freezing point (Fig. 19b).

The difference between the freezing temperature under usual (equilibrium) conditions and the freezing temperature when the liquid metal is cooled at a high rate is called *the degree of supercooling*:

$$\Delta t = t_e - t_s \quad (45)$$

where:  $\Delta t$  = degree of supercooling,

$t_e$  = equilibrium freezing temperature,

$t_s$  = freezing temperature at a high cooling rate.

Amorphous solids do not have a horizontal line in their cooling curves and gradually pass from the liquid to the solid state as the temperature falls (Fig. 19c).

G. Tammann established a definite relationship between the rate of nucleation, the rate of crystal growth and the degree of supercooling.

This relationship is represented graphically in Fig. 20. Upon an increase in the degree of supercooling  $\Delta t$ , both the rate of nucleation

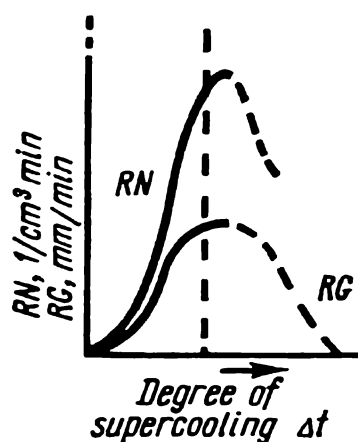


Fig. 20. Effect of the degree of supercooling on the rates of nucleation (RN) and linear crystal growth (RG)

$RN$  and the rate of crystal growth  $RG$  first increase and then drop again. At a low degree of supercooling in the molten metal, the rate of nucleation is low and the nuclei grow slowly. Under such freezing conditions large crystals should form in the metal.

At a higher degree of supercooling, when the rate of nucleation approaches its maximum value and the rate of crystal growth has already passed its maximum and has decreased slightly, small crystals of the metal mainly form.

The degree of supercooling depends upon the nature of the metal and the cooling rate. Certain metals, for example tin, antimony and bismuth, have a greater susceptibility to supercooling than other metals.

The freezing of a metal depends to a great extent upon the rate at which heat is removed from the melt. As the nuclei are formed (nucleation) the chaotically arranged atoms of the liquid metal begin to arrange themselves in a definite order around the nuclei, forming microscopic crystals. Since freezing begins simultaneously at many

points in the volume of metal, crystal growth in the form of perfect geometrical bodies is gradually impaired due to interference of each crystal with its neighbours. This leads to the formation of a great number of irregular crystals from which the granular structure of metals is obtained. Almost always, under actual conditions, crystalline grains in the form of *dendrites* develop in the freezing of a liquid metal (*dendron* is the Greek for a tree).

A schematic diagram showing the formation and growth of a dendrite is given in Fig. 21a. As can be seen in the diagram, when the liquid metal solidifies near one of the nuclei, first the main (long) branch of the dendrite is formed. It is usually called an axis of the first order. Axes of the second order branch off at a definite angle

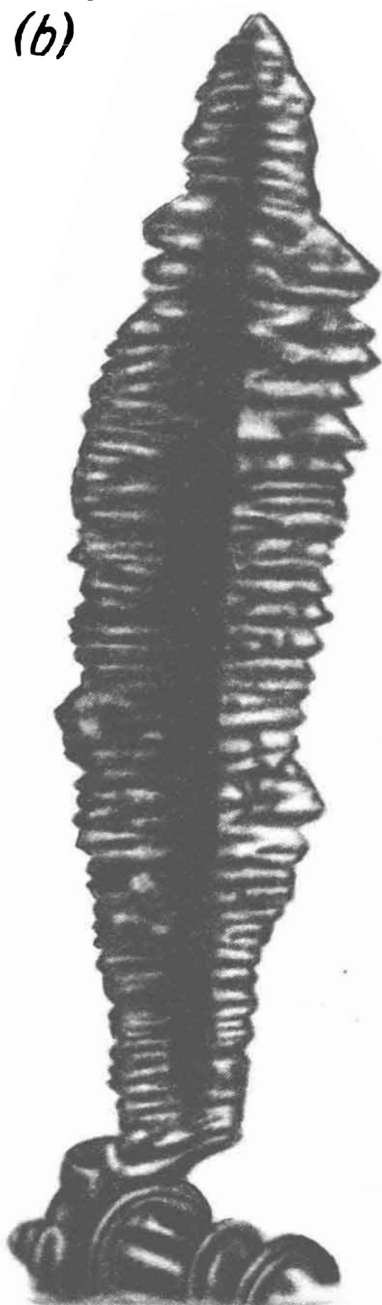
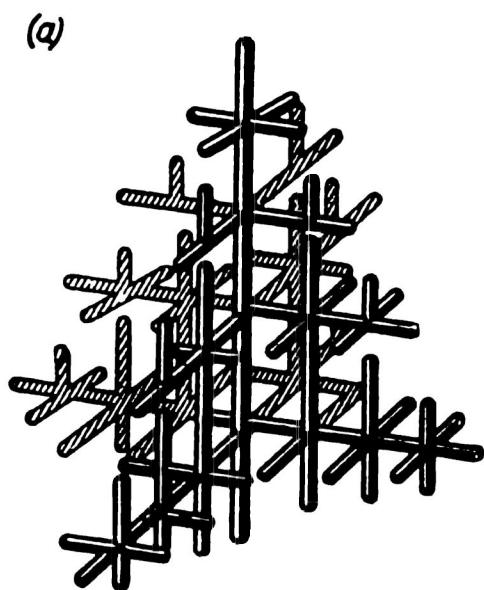


Fig. 21. Dendrites:  
(a) dendritic growth; (b) D. Chernov's dendrite

from the axis of the first order. Axes of the third order branch off from the second-order axes and so forth.

D. Chernov's famous crystal (dendrite) which he found in the pipe of a large steel ingot is shown in Fig. 21b.

If a metal is subjected to an allotropic transformation in the solid state then recrystallisation will be observed in the metal as it cools. Recrystallisation involves a rearrangement of the crystal lattice and the formation of new crystals from the crystals of the previous

arrangement. Recrystallisation is accompanied by the evolution of heat and is characterised by a constant-temperature interval (horizontal line) on the cooling curve.

## 5-2. Alloys

Pure metals are comparatively seldom used; in engineering, application is made chiefly of alloys which consist of two or more metals, or of metals and metalloids.

*Alloys* are metallic solids, complex in composition, formed as a result of the freezing of the melt—the liquid solution of two or more metals, or metals and metalloids.

Each constituent of an alloy is called a *component*. Alloys may be binary (two-component), ternary (three-component), etc.

The ability of various metals to form alloys differs greatly and, therefore, the structure of various alloys after solidification may also be very diverse.

In the liquid state, alloys are entirely homogeneous and from the physical point of view constitute a single phase \*. Nonhomogeneity may appear when an alloy is transformed from the liquid to the solid state, i.e., several solid phases are formed. After solidification, alloys may consist of one, two or more phases depending upon the nature of their components. Certain metals are not mutually soluble in the liquid state; upon being fused they form two layers with different specific weights (e.g., lead and iron, lead and zinc, etc.). It is difficult to form an alloy in such cases since it is necessary to mix the metals into each other.

Complete insolubility of metals in the liquid state, however, is quite rare; limited solubility is more frequently observed. If the concentration of one of the metals exceeds its maximum solubility in the other metal, the liquid will also separate into two layers thereby impeding the making of the alloy. A limited solubility in the liquid state is most frequently found in metals whose atomic volumes and melting points differ substantially from each other.

After solidification, the components of an alloy may form: (1) solid solutions, (2) chemical compounds, or (3) mechanical mixtures of crystals.

**Solid solutions.** In solid solutions the crystal lattice of the alloy is formed of different types of atoms belonging to the two components.

*The solvent* is the name given to the component whose crystal lattice is retained in the formation of the solid solution. If the two metals have crystal lattices of the same type and are completely

---

\* A *phase* is a homogeneous portion of a nonhomogeneous system separated from the other portions by the interface.

mutually soluble in the solid state, the solvent is the component whose concentration in the alloy exceeds 50 atomic per cent.

Conditions under which metallic solid solutions are formed are: (1) the components have the same type of crystal lattice and (2) there is no large difference in the lattice constants of the alloying components.

Diffusion processes may take place in solid solutions as well as in liquid ones. Particles may pass from volumes of higher concentration in a given solution to volumes of lower concentration until the concentration in various places of the solution becomes the same.

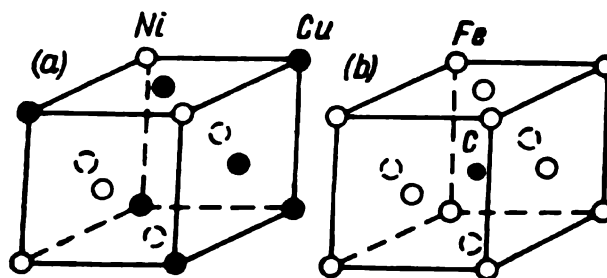


Fig. 22. Unit cells of crystal lattices of solid solutions:

(a) substitutional; (b) interstitial solid solutions

Diffusion in solid solutions, however, is considerably slower than in liquid solutions; the rate of diffusion increases with temperature.

The two most typical kinds of solid solutions are substitutional and interstitial.

In *substitutional solid solutions* (Fig. 22a) certain of the atoms at the points of the solvent crystal lattice have been replaced by solute atoms. When *interstitial solid solutions* are formed, the atoms of the solute metal occupy the vacant positions or interstices between the solvent atoms (Fig. 22b).

**Chemical compounds.** Chemical compounds are obtained upon chemical interaction of certain components of alloys. These compounds are characterised by a special type of crystal lattice which, in most cases, differs from those of the components. Chemical compounds in an alloy not only have a definite chemical composition but also possess distinct physical and mechanical properties (high hardness, higher brittleness and high electrical resistivity).

Chemical compounds in alloys may be formed either by metals (intermetallic compounds) or by a metal and a metalloid. Certain compounds of metals and metalloids such as, for example, carbides, nitrides, oxides, phosphides, etc., have found independent applications in engineering.

**Mechanical mixtures.** In some cases when a liquid solution is cooled, crystals of all its components separate simultaneously from the melt and form a close mechanical mixture (eutectic).

Investigations have shown that the crystals constituting a mechanical mixture retain the crystal lattices of the original components of the alloy.

Many alloys have a very complex structure. They may simultaneously contain structural constituents in the form of mechanical mixtures of crystals of the alloy components, chemical compounds of these components and solid solutions.

5-3. Main Types of Equilibrium Diagrams of Binary Alloys

Equilibrium diagrams, also called constitutional or phase diagrams, represent the generalised results of investigations in the solidification process and the structural and phase transformations of the given alloy system. These diagrams enable the temperatures to be determined at the beginning and end of melting for alloys of various composition, the structures of the alloys to be found for various temperatures under equilibrium conditions and also the transformations the alloys are subjected to during cooling and heating to be determined.

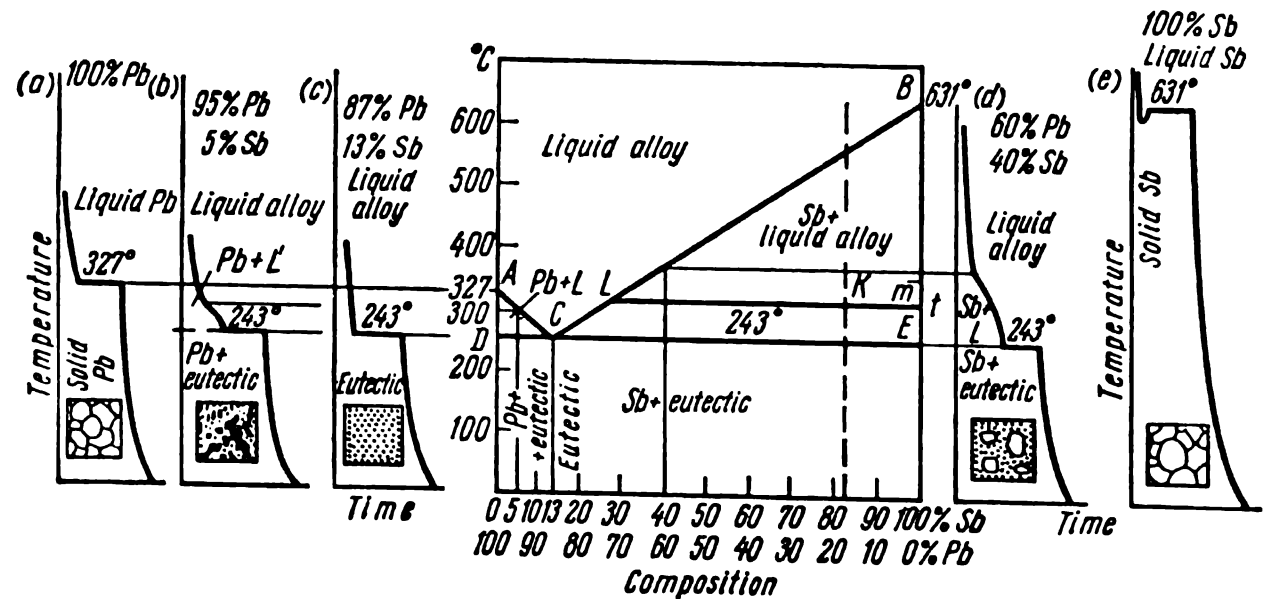


Fig. 23. Cooling curves, structures and the equilibrium diagram of lead-antimony alloys

An equilibrium diagram is plotted by laying off the percentage concentrations of the two components (from 0 to 100 per cent) along the axis of abscissas and the temperatures along the axis of ordinates. Any point on the diagram refers to a definite composition of the alloy at a definite temperature.

Various methods of physico-chemical analysis are employed to obtain the data necessary to draw equilibrium diagrams of alloys

and to investigate their structure. Such methods include thermal, microscopic, X-ray structural analysis and others.

Thermal analysis is used to determine the temperatures of the beginning and end of freezing when an alloy passes from the liquid to the solid state, as well as the temperatures of all the transformations that occur in the alloy in the solid state. This method involves measurement of the temperature of the alloy being cooled at equal intervals of time.

To make a thermal analysis, a series of alloys are prepared with a gradually varying concentration of one of the component metals. Alloys are taken, for example, containing 10, 20, 30, 40 per cent, etc., of the same component. A series of such alloys is heated until they melt and then are slowly and uniformly cooled. The temperature of the alloy is measured with a thermocouple after definite intervals of time. On the basis of the data obtained a series of cooling and heating curves are plotted in temperature vs. time coordinates. The points of these curves are then used to draw the equilibrium diagram.

**Equilibrium diagram of a binary system in which the components are completely soluble in the liquid state and completely insoluble in the solid state, forming a mechanical mixture of the crystals.** Equilibrium diagrams of this type are valid for the metal systems Pb-Sb, Sn-Zn, Pb-Ag and others.

As an example we shall construct an equilibrium diagram for the Pb-Sb system of alloys.

The melting point of lead is 327° C; that of antimony is 631° C. If we take several Pb-Sb alloys of different composition with a gradually increasing Sb content and cool them slowly from the liquid to the solid state, the following temperatures will be observed for the beginning and end of solidification (Table 1).

*Table 1*

**Freezing Temperatures of Alloys in the Pb-Sb System**

Alloy	Per cent composition		Freezing temperature, °C	
	Sb	Pb	beginning	end
I	5	95	296	243
II	10	90	260	243
III	13	87	243	243
IV	20	80	280	243
V	40	60	395	243
VI	80	20	570	243

Fig. 23 illustrates the cooling curves for pure lead and antimony and certain of their alloys. The cooling curves for the pure metals (Fig. 23a and e) have only one step (horizontal line) corresponding



to their freezing point. On each of the cooling curves of the alloys in the Pb-Sb system (Fig. 23*b* and *d*) two characteristic inflexions are observed which represent the beginning (upper point) and end (lower point) of solidification. The upper points at which alloys of various composition in the system begin to freeze differ substantially in temperature; the lower critical points are located at the same temperature (243° C) for all alloys of lead and antimony.

Projecting the points obtained for the beginning and end of solidification on the alloy composition vs. temperature coordinate system, we can plot the equilibrium diagram for the Pb-Sb system.

Lead-antimony alloys of any composition are in the liquid state above the lines *AC* and *CB*. The lines *AC* and *CB* of this diagram represent the beginning of solidification of the alloys and are called the *liquidus lines* (from the Latin *liquidus* meaning to be fluid). The straight line *DCE* indicates the end of solidification of all the alloys in the system and is called the *solidus* (also from the Latin). Below this line lead-antimony alloys of any composition are in the solid state. Pure lead begins to separate from the melt of alloys containing less than 13% Sb on line *AC*. Pure antimony begins to separate on line *BC* from alloys containing more than 13% Sb.

At point *C* which corresponds to an alloy containing 13% Sb and 87% Pb both lead and antimony will simultaneously separate from the melt at a temperature of 243° C, and their crystals will form a mechanical mixture. The structure obtained after the solidification of this alloy is called the *eutectic* and the alloy corresponding to point *C* is called the *eutectic alloy*. For a given system of components the eutectic alloy is characterised by its constant composition and the lowest melting point of all the alloys in the system (other Pb and Sb alloys in this case).

All alloys whose compositions on the diagram are located to the left of point *C* are called *hypoeutectic alloys*; those to the right are called *hypereutectic alloys*.

A hypoeutectic alloy (for example, the one in Fig. 23*b*), containing less than 13% Sb, is in the liquid state above line *AC*; upon cooling below line *AC* crystals of pure lead begin to separate from the liquid alloy. These lead crystals have a higher freezing point than the remaining liquid part of the alloy which gradually becomes richer in antimony. The separation of pure lead crystals from the liquid phase continues to the temperature of 243° C, i.e., to the eutectic line *DCE*. At this temperature the remaining liquid phase contains 13% Sb and it will freeze as a whole to form the eutectic.

A somewhat different picture is observed when hypereutectic alloys containing more than 13% Sb (Fig. 23*d*) are cooled. Here crystals of pure antimony begin to separate from the liquid alloy below line *BC*. Antimony has a higher freezing point than the remaining liquid

part of the alloy; therefore, the latter is depleted of antimony and its composition will approach that of the eutectic. At a temperature corresponding to line  $DCE$  ( $243^\circ\text{C}$ ) the remaining liquid part of the alloy contains 13% Sb. Upon a further drop in temperature this liquid part freezes as a whole and forms the eutectic mixture.

Thus, hypereutectic alloys solidify in the same manner as hypoeutectic alloys except that antimony, and not lead, is separated from the melt at temperatures below the liquidus.

The eutectic alloy containing 13% Sb and 87% Pb solidifies at a strictly constant temperature (point  $C$  at  $243^\circ\text{C}$ ) with crystals of both lead and antimony separating from the melt simultaneously to form the eutectic (Fig. 23c).

**Lever rule.** In addition to determining the regions in which various phases exist and their qualitative composition at a given temperature, an equilibrium diagram may be used to find the quantitative relationship between the separate phases. The lever rule serves this purpose.

If we wish to determine the quantitative ratio between the phases at point  $k$  (temperature  $t$ ) in Fig. 23, we draw a horizontal line  $lm$  through point  $k$ ; this line will be parallel to the axis of abscissas. Then point  $m$  shows the composition of the solid phase, consisting of 100% Sb, and point  $l$  shows the composition of the liquid phase in the alloy.

Denoting the weight of the whole amount of the alloy by  $Q$ , the weight of the liquid phase by  $Q_L$  and that of the solid phase by  $Q_S$ , we can, using the lever rule, write the following equations:

$$\frac{Q_S}{Q} = \frac{lk}{lm}; \quad \frac{Q_L}{Q} = \frac{km}{lm} \quad \text{and} \quad \frac{Q_S}{Q_L} = \frac{lk}{km}.$$

**Equilibrium diagram of a system in which the components are completely soluble in the liquid state, insoluble in the solid state and form a stable chemical compound.** In discussing this type of diagram we shall consider a system with the components  $A$  and  $B$  which combine to form the stable chemical compound  $A_mB_n$  (Fig. 24).

The chemical compound  $A_mB_n$  should be regarded as an independent component in the diagram having a melting point either higher or lower than those of the original components  $A$  and  $B$ . In this connection, the equilibrium diagram should be regarded as comprising two systems: (1)  $A_1-A_mB_n$  and (2)  $A_mB_n-B_1$  each of which is a typical diagram of the first type.

In the system  $A_1-A_mB_n$  the beginning of solidification is indicated by the line  $A_1E_1C_1$ . Component  $A$  separates from the melt along the line  $A_1E_1$  and the chemical compound  $A_mB_n$  along the line  $E_1C_1$ . Final solidification of the alloy occurs along the eutectic temperature line  $D_1E_1F_1$ . Crystals of  $A$  and  $A_mB_n$  separate simultaneously from

the melt at point  $E_1$ ; an alloy of this composition has a eutectic structure.

Alloys of the system  $A_mB_n$ - $B_1$  begin to solidify along the line  $C_1E_2B_1$ , the chemical compound  $A_mB_n$  being separated along  $C_1E_2$  and the pure component  $B$ —along the line  $E_2B_1$ . These alloys completely solidify at the eutectic temperature (line  $KE_2L_1$ ). The eutectic alloy corresponding to point  $E_2$  consists of  $A_mB_n + B$ .

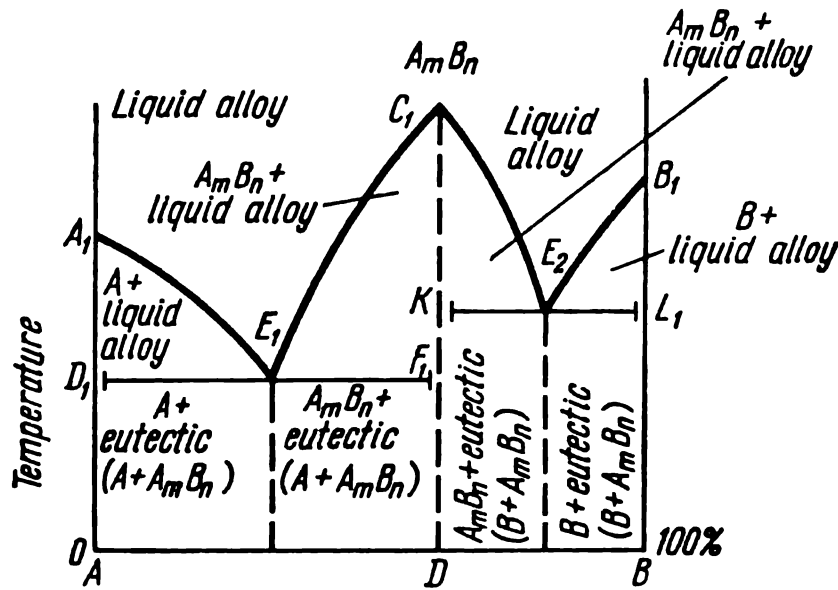


Fig. 24. Equilibrium diagram of binary alloys whose components form a chemical compound

Alloys such as magnesium-tin, magnesium-lead, cobalt-antimony, manganese-silicon and others solidify according to this type of diagram.

**Equilibrium diagram of a system in which the components are completely soluble in both the liquid and solid states, and form homogeneous solid solutions.** Complete mutual solubility is observed in systems such as Cu-Ni, Fe-Ni, Co-Cr and certain others.

Fig. 25 illustrates the equilibrium diagram for the Cu-Ni system.

The melting point of pure copper is  $1083^{\circ}\text{C}$ ; that of nickel is  $1452^{\circ}\text{C}$ . At temperatures above the line  $ACB$  (liquidus) alloys of all compositions are in the liquid state; below the line  $ADB$  (solidus) they are all in the solid state. Between the liquidus and solidus, alloys of any composition are partly in the liquid and partly in the solid state.

As an example, we may consider an alloy containing 50% Cu and 50% Ni.

At a temperature corresponding to the point  $a$  on the liquidus ( $ACB$ ), the first crystals of the solid solution of Cu and Ni begin to separate from the liquid phase. The composition of this solid solution is determined by point  $a_1$  on the line  $ADB$ . It is evident from the

diagram that these crystals contain more nickel than the initial liquid phase. Upon a further drop in temperature, the amount of solid solution crystals increases and their composition changes along the solidus in the direction of the arrow, approaching the composition of the original alloy, i.e., becomes leaner in nickel.

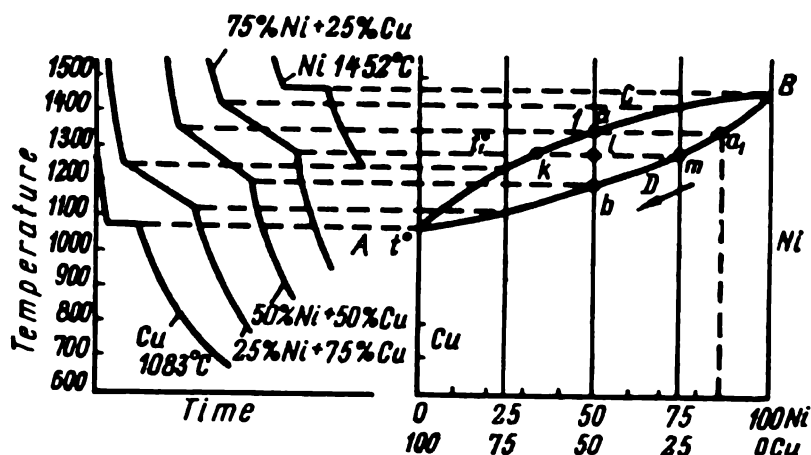


Fig. 25. Equilibrium diagram of binary alloys whose components are completely mutually soluble in both the liquid and solid states

In the course of solidification the amount of liquid phase is gradually reduced and the liquid itself becomes richer in copper. The alloy is completely solid when its temperature reaches the point  $b$  on the solidus  $ADB$ .

Using the lever rule, the quantities of the liquid and solid phases in the temperature interval between points  $a$  and  $b$  may be readily determined. Thus, at temperature  $t_1$ , point  $m$  on the horizontal line  $klm$  indicates the composition of the solid phase and point  $k$ , that of the liquid phase. In the given case, the ratio of the amounts of solid and liquid phases is

$$\frac{Q_s}{Q_L} = \frac{kl}{ml}.$$

Equilibrium diagram of a system in which the components are completely soluble in the liquid state and have limited solubility in the solid state. An equilibrium diagram for components  $A$  and  $B$  which have only limited solid solubility is shown in Fig. 26. Point  $D$

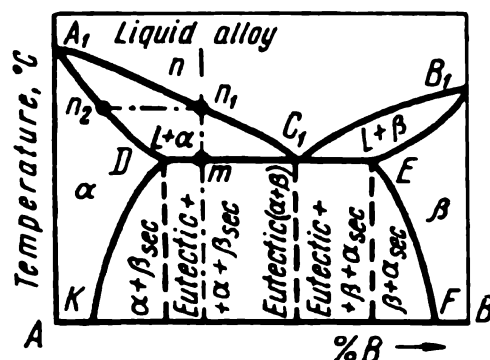


Fig. 26. Equilibrium diagram of binary alloys whose components are completely soluble in the liquid state and have limited solubility in the solid state

indicates the maximum solubility of component  $B$  in component  $A$  (in the alpha solid solution). Point  $E$  corresponds to the maximum solubility of component  $A$  in component  $B$  (in the beta solid solution). For this reason, not pure components  $A$  and  $B$  separate from the

melt along the liquidus  $A_1C_1B_1$ , but crystals of the alpha solid solution of component  $B$  in component  $A$  (line  $A_1C_1$ ) and crystals of the beta solid solution of component  $A$  in component  $B$  (line  $C_1B_1$ ).

The eutectic formed in the freezing of these alloys (point  $C_1$ ) is a fine-grained mixture of crystals of the alpha and beta solid solutions whose compositions correspond to points  $D$  and  $E$ .

As an example, let us consider the solidification of a hypoeutectic alloy  $n$ .

The alloy is in the liquid state above the line  $A_1C_1$ . As the alloy cools the first crystals of the alpha solid solution begin to separate at point  $n_1$  on the line  $A_1C_1$  and their composition corresponds to point  $n_2$ . Upon further cooling, the composition of the solid crystals that separate from the liquid phase will vary along the line  $n_2D$ , while that of the liquid alloy—along the line  $n_1C_1$ .

Below the liquidus  $A_1C_1B_1$  and above the solidus  $A_1DC_1EB_1$  the solidifying alloy comprises two phases—liquid and solid. When the temperature of the alloy reaches point  $m$ , a eutectic is formed of the remaining liquid. This eutectic consists of crystals of alpha and beta solid solutions with compositions corresponding to points  $D$  and  $E$ , respectively. Slightly below point  $m$  the alloy consists of the previously separated crystals of alpha solid solution surrounded by the eutectic. Upon further cooling, the composition of the solid solution changes along the line  $DK$  and it is said to decompose, i.e., secondary crystals of the beta solid solution, which become leaner in component  $B$ , separate from the alpha solid solution. These secondary crystals are called the eutectoid.

A similar phenomenon is observed in hypereutectic alloys where the beta solid solution decomposes along the line  $EF$ .

#### 5-4. Effect of the Composition of Alloys on Their Properties

Equilibrium diagrams enable alloys to be properly selected for various purposes and their behaviour to be predicted to some degree when they are subjected to various processing operations (casting, heat treatment, etc.). Thus, for example, alloys which do not undergo allotropic transformations are not capable of forming solid solutions with variable composition of the components upon freezing. Eutectic alloys have the lowest freezing point in the given system of alloys and thereby possess good castability, i.e., casting properties.

The fine-grained structure of the eutectic imparts good mechanical properties to the alloy.

The relationship between the composition, structure and properties of alloys was first established by Academician N. Kurnakov.

Certain properties of alloys which form a eutectic in freezing (electrical resistivity and tensile strength, in the given case) change

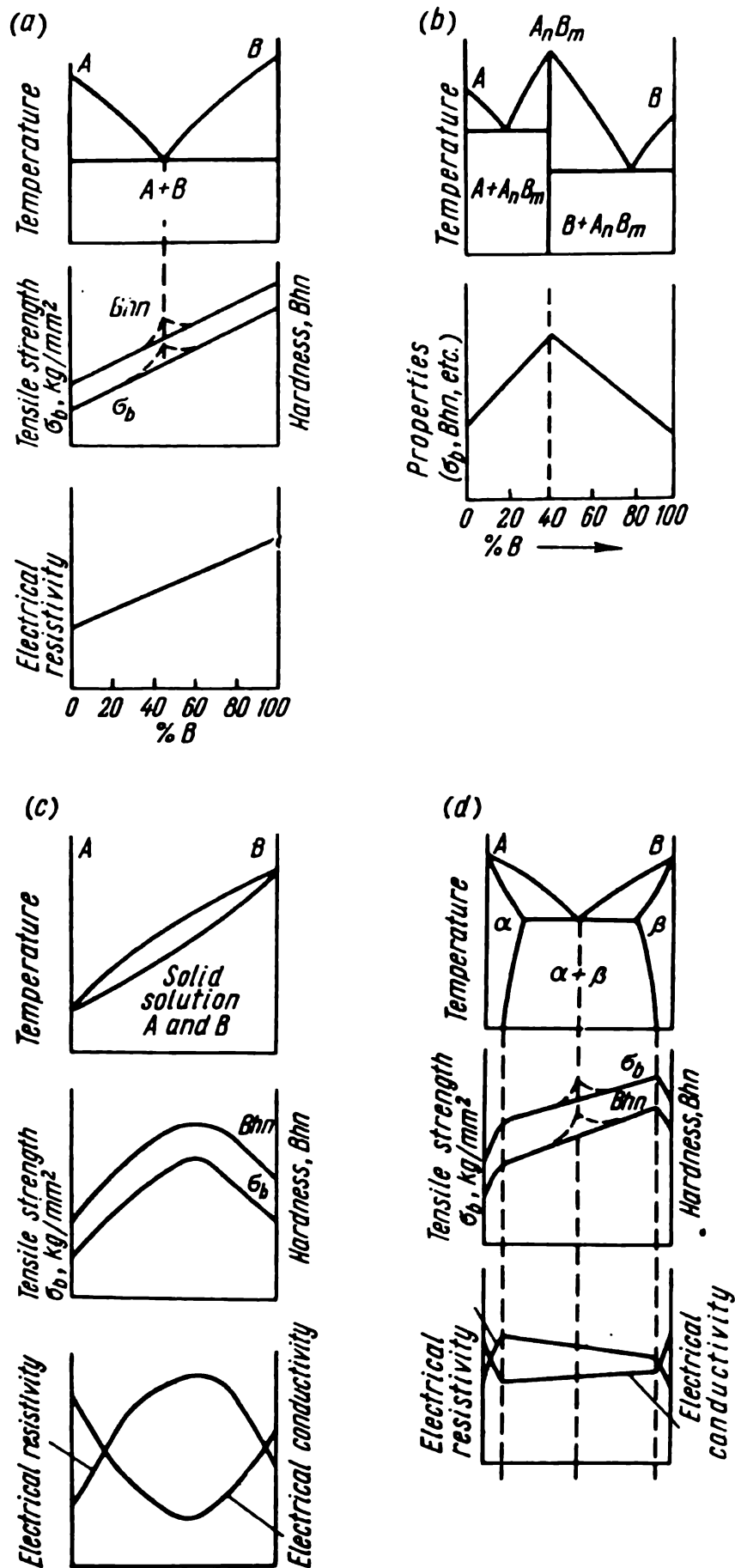


Fig. 27. Relation between equilibrium diagrams and the properties of the alloys they represent.

Types of alloys: (a) with a eutectic; (b) forming a stable chemical compound; (c) forming a continuous series of solid solutions; (d) with limited solid solubility

gradually from the value of the property for one component to that for the other (Fig. 27a).

If a chemical compound  $A_nB_m$  is formed in alloys the properties vary linearly from component  $A$  to the chemical compound  $A_nB_m$  and from there to component  $B$  (Fig. 27b). A sharply defined inflexion is observed on the properties vs composition diagram at the point corresponding to the composition of the chemical compound  $A_nB_m$ ; it is called the *singular point*. Properties vary along a curve from  $A$  to  $B$  for alloys which form a continuous series of solid solutions. For example, the hardness Bhn, tensile strength  $\sigma_b$  and electrical resistivity are increased when solid solutions are formed while the electrical conductivity is reduced (Fig. 27c). The properties of alloys with limited solid solubility vary along curves in the regions of homogeneous solid solutions and along a straight line in the two-phase region (Fig. 27d).

### 5-5. Diagrams of Ternary Systems

Alloys containing not only two but three, four and even more components have found wide application in industry. At the present time many three- and some four-component alloys have been comprehensively investigated. The diagram of a ternary system is

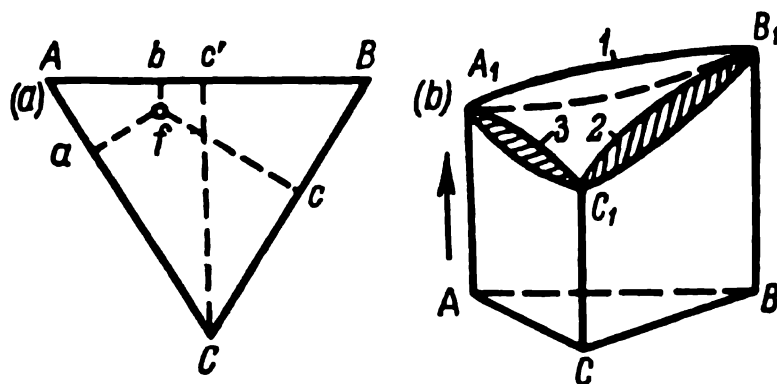


Fig. 28. Construction of a ternary diagram

a three-dimensional model (Fig. 28). The base of the model is an equilateral triangle on whose sides the concentrations of the components are plotted. This gives it its name—the concentration triangle. The apexes of the triangle represent the three pure components  $A$ ,  $B$  and  $C$ . Perpendiculars are erected from the apexes of the triangle and along them the temperature values  $t$  are plotted. The binary alloys of  $A$  and  $B$  are represented by line  $AB$ ; those of  $B$  and  $C$  by line  $BC$  and those of  $A$  and  $C$  by line  $AC$ . Any point within the triangle specifies the composition of a ternary alloy.

The composition of ternary alloys may be determined on the basis of one of the properties of an equilateral triangle. This geometrical

rule states that the sum of three lines drawn perpendicular to the three sides from any point within the triangle is a constant value equal to the altitude of the triangle, e.g., for point  $f$  (Fig. 28a)  $fa + fb + fc = Cc'$ . Taking the altitude as 100 per cent, the composition of the alloy, i.e., the content of the three components in per cent may be determined as

$$A = \frac{fc}{Cc'} 100; \quad B = \frac{fa}{Cc'} 100; \quad C = \frac{fb}{Cc'} 100. \quad (46)$$

The first step in constructing a ternary diagram is to plot the cooling curves for alloys of various composition to determine the critical points. Then perpendiculars are erected to the plane of the triangle at the corresponding points for each alloy investigated and the temperatures of the critical points are plotted on the corresponding perpendicular. The simplest ternary diagram is one for a system in which all three components are mutually soluble in the solid state. Such a diagram is shown in Fig. 28b. The convex surface  $A_1B_1C_1A_1$  indicates the beginning of freezing of the alloys and is, in fact, the liquidus surface. The concave surface  $A_1B_1C_1A_1$ , below the liquidus, is the solidus surface and indicates the temperatures at which the alloys are completely solidified. Above the liquidus all alloys are in the liquid state; below the solidus they are all in the solid state. Between these two surfaces all alloys are partly liquid and partly solid, i.e., they consist of crystals of solid solution of the three components,  $A$ ,  $B$  and  $C$ , and a certain amount of liquid. After complete solidification all the alloys of this system consist of crystalline grains of the solid solution of the three components.

The practical value of ternary diagrams is that they enable the temperatures at the beginning and end of solidification to be determined for ternary alloys, as well as the temperature conditions for their hot working and heat treatment procedures.

## Chapter 6

### MECHANICAL PROPERTIES OF METALS AND ALLOYS

*The mechanical properties* of metals are determined by experiments conducted on test pieces or finished articles.

Mechanical tests of metals may be: *static*, when the test piece is subjected to the action of a constant or slowly increased load; *dynamic*, when the load increases instantaneously, as in an impact, and *cyclic* or *fatigue tests*, when the load repeatedly varies in magnitude and direction in the course of the test.



The most extensively employed mechanical tests for metals are: (1) tension tests, (2) hardness tests, (3) impact and fatigue tests and (4) fabrication tests (bending, jumping-up, flattening and others).

6-1. Tension Tests for Metals and Alloys

Standard specimens of either round or rectangular cross section are used for tension tests (Fig. 29).

Standard tension specimens have a cross-sectional area of 314 sq mm ( $d=20$  mm). Two types are employed: the long specimen in which the gauge length is 200 mm (the gauge length to diameter ratio is 10) and the short specimen with a gauge length of 100 mm (the gauge length to diameter ratio is 5).

The cross-sectional area of standard specimens is selected arbitrarily, but the gauge length must satisfy the following conditions for long and short specimens, respectively:

$$l_0 = 11.3 \sqrt{F_0} \text{ and } l_0 = 5.65 \sqrt{F_0}, \tag{47}$$

where  $F_0$  is the initial cross-sectional area of the specimen in sq mm.

In a tension test the round or rectangular specimen is deformed by the action of a smoothly increasing load up to the point of rupture. During the test, readings are made to obtain data for plotting a tension-test diagram. Such a diagram shows the relation between the force applied to the specimen and its resulting deformation (Fig. 30).

On the diagram, the values of the applied load  $P$  are plotted as ordinates and the absolute elongation  $\Delta l$  corresponding to each load is plotted as an abscissa.

On section  $OP_p$  of the curve, the elongation  $\Delta l$  of the specimen is directly proportional to the load. This relation agrees with the law of proportionality which states that variation in the load applied to the specimen will lead to a proportional variation in the strain.

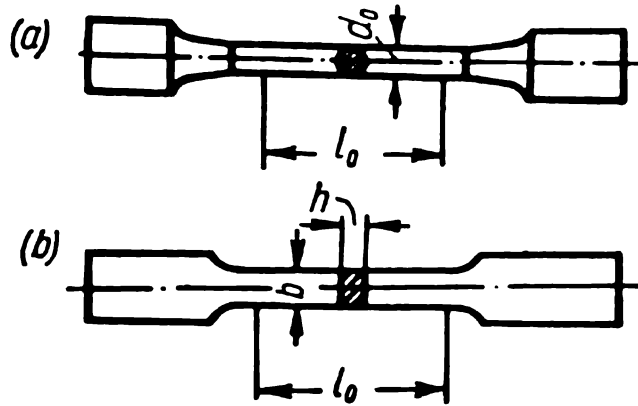


Fig. 29. Standard specimens used in tension tests:  
(a) round; (b) rectangular  
Principal dimensions:  $d_0$ —diameter of the round specimen;  $h$ —thickness and  $b$ —width of the rectangular specimen;  $l_0$ —gauge length of the specimen

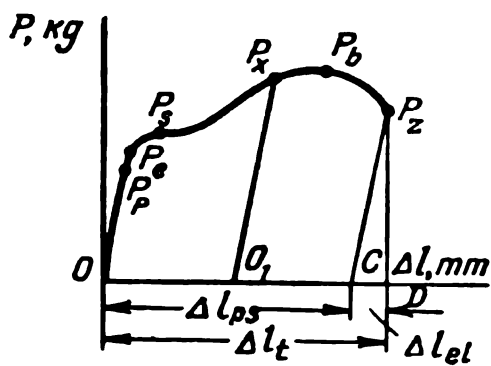


Fig. 30. Tension diagram for low-carbon steels

The load  $P_p$ , at which a straight-line relation is still observed between load and elongation, is called *the proportional limit load*.

*The proportional limit*  $\sigma_p$  is the maximum stress (load per square millimetre of the initial cross-sectional area) at which the relative elongation remains directly proportional to the load:

$$\sigma_p = \frac{P_p}{F_0} \text{ kg per sq mm.} \quad (48)$$

The load  $P_e$  at which the permanent set of the specimen is equal to 0.5 per cent of its initial gauge length is called *the elastic limit load*. The load  $P_e$  is very near in value to the load  $P_p$ .

*The elastic limit*  $\sigma_e$  is defined as the stress at which the permanent set is equal to 0.5 per cent of the initial gauge length of the specimen:

$$\sigma_e = \frac{P_e}{F_0} \text{ kg per sq mm.} \quad (49)$$

Above point  $P_e$  the tension curve deviates considerably from a straight line and at a load corresponding to  $P_s$  goes over into a horizontal section. In this region the material of the specimen receives considerable permanent set without an increase in the applied tensile load; the specimen seems to cease offering resistance to the tensile forces, i.e., it flows.

The load  $P_s$  at which the material begins to flow is called *the yield point load* and the horizontal part of the curve where the specimen elongates without increasing the load is called *the yield step*.

*The yield point*  $\sigma_s$  is the minimum stress at which the specimen is deformed without a noticeable increase in load:

$$\sigma_s = \frac{P_s}{F_0} \text{ kg per sq mm.} \quad (50)$$

Beyond the yield step the load can be increased again to a certain maximum value  $P_b$  where a localised reduction of cross-sectional area, or necking down, occurs. This constriction of area at the neck leads to a drop in the load, and at point  $P_z$  the specimen is ruptured.

The maximum load  $P_b$  after which necking down begins is called *the tensile strength load*.

*The tensile, or ultimate, strength* is the stress corresponding to the maximum load reached before rupturing the specimen:

$$\sigma_b = \frac{P_b}{F_0} \text{ kg per sq mm.} \quad (51)$$

The total strain (deformation)  $\Delta l_t$  (elastic and permanent at the moment of fracture) is equal to the distance  $OD$  (see Fig. 30).

To obtain the separate values of the elastic strain and the permanent set at the moment of fracture it is necessary to draw a straight

line on the tension diagram from point  $P_z$  to the axis of abscissas, parallel to the straight-line section of the load-strain curve. The line  $OC$  represents the amount of permanent set  $\Delta l_{ps}$  and the line  $CD$ —the total elastic strain.

The true tensile strength  $\sigma_z$  is the stress in the neck of the specimen and equals the ratio of the load, applied to the specimen directly before rupture, to the cross-sectional area of the neck:

$$\sigma_z = \frac{P_z}{F_1} \text{ kg per sq mm,} \quad (52)$$

where:  $P_z$  = load measured immediately before fracture of the specimen,

$F_1$  = cross-sectional area of the neck of the specimen after its rupture.

The *relative elongations*  $\delta$  is the ratio of the length increment of the specimen after fracture to its initial gauge length, expressed in per cent:

$$\delta = \frac{l_1 - l_0}{l_0} 100\%, \quad (53)$$

where:  $l_1$  = length of the specimen after rupture, in mm,

$l_0$  = initial gauge length of the specimen, in mm.

The *relative reduction in area*  $\psi$  is the ratio of the maximum reduction in cross-sectional area of the specimen after rupture to the initial area expressed in per cent:

$$\psi = \frac{F_0 - F_1}{F_0} 100\%, \quad (54)$$

where:  $F_0$  = cross-sectional area of the specimen, in sq mm, before the test,

$F_1$  = cross-sectional area, in sq mm, at the fracture.

## 6-2. Determining the Hardness of Metals and Alloys

**Steel ball indentation hardness tests.** The most common method of measuring the hardness of metals is by determining the resistance offered to the indentation of a hardened steel ball.

As a result of this penetration, a spherical impression remains on the surface of the metal (Fig. 31a).

The hardness number determined by this method is denoted by Bhn (Brinell hardness number) and is defined as the pressure in kilograms per square millimetre of the spherical surface of the impression:

$$\text{Bhn} = \frac{P}{F} \text{ kg per sq mm,} \quad (55)$$

where  $F$  is the area of the spherical surface of the impression in sq mm.

The harder the metal, the higher its Brinell number will be.

**Determining hardness by the depth of indentation of a diamond cone or a small hardened steel ball.** Frequently a Rockwell test is applied to determine the hardness of a metal. This is based on the indentation of a hard tip, or indenter, into the test specimen under the action of two consecutively applied loads—minor (initial) and major (final) (Fig. 31b).

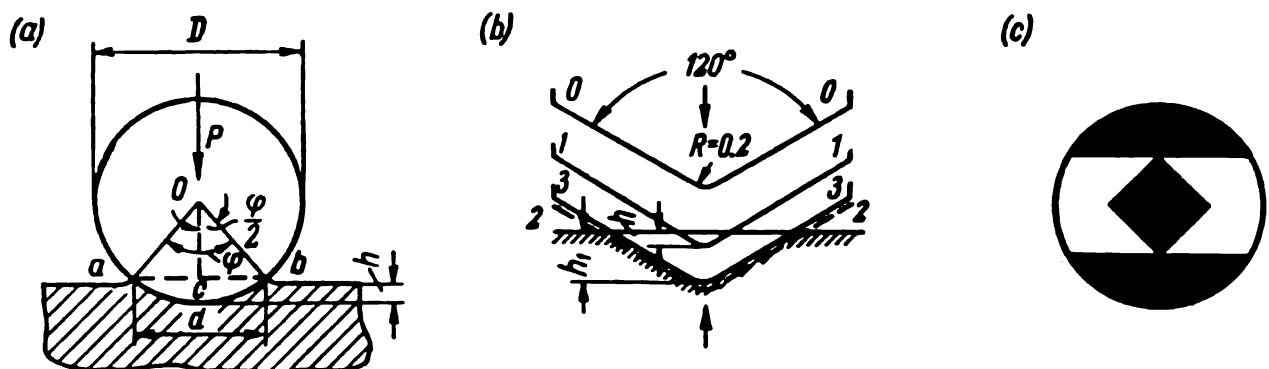


Fig. 31. Hardness tests:

(a) by indentation of a steel ball (Brinell principle); (b) by indentation of a diamond cone (Rockwell principle); (c) by indentation of a diamond pyramid (Vickers principle); 1-1—penetration of the cone under the minor load  $P_0$ ; 2-2—penetration of the cone under the major (final) load  $P$ ; 3-3—penetration of the cone after the major load  $P$  is reduced again to the value of the minor load  $P_0$ .

The indenter, or “penetrator”, in a Rockwell test for hard materials is a conical-shaped diamond (called a brale) with a  $120^\circ$  apex angle; for softer materials, a hardened steel ball 1.5875 mm ( $1/16$ ”) in diameter is used.

The minor (initial) load is 10 kg; the major load (minor + additional) when testing with the brale is 150 kg (Scale C) or 60 kg (Scale A). The major load for the ball indenter is 100 kg (Scale B).

The Rockwell hardness number is expressed in conventional units computed by the formula

$$R = \frac{K - (h_1 - h)}{C}, \quad (56)$$

where:  $R$  = Rockwell hardness number,  
 $K$  = a constant equal to 0.26 for the steel ball and 0.2 for the brale,  
 $h_1$  = depth of penetration, in mm, of the indenter under the action of the major load  $P$  but measured after reducing it to the minor load  $P_0$ ,  
 $h$  = depth of penetration, in mm, of the indenter under the action of the minor load  $P_0 = 10$  kg,  
 $C$  = value of the dial indicator scale divisions for an indenter penetration of 0.002 mm.

A subindex added to the hardness symbol  $R$  indicates the scale on which the test was conducted; for example:  $R_A$ ,  $R_B$  and  $R_C$  refer to scales  $A$ ,  $B$  and  $C$ , respectively.

**Determining hardness by indentation of a diamond pyramid.** The test consists in indentation of a square-based diamond pyramid having an angle of  $136^\circ$  between opposite faces. The applied load is 5, 10, 30, 50, 100, or 120 kg. The impression produced by the test is of the form shown in Fig. 31c. The load is selected according to the thickness and hardness of the specimen.

The Vickers hardness number, or diamond pyramid hardness (DPH), determined in such tests is the load per unit area of the impression and is found from the formula:

$$\text{DPH} = \frac{2P \sin \frac{\alpha}{2}}{d^2} \text{ kg per sq mm,} \quad (57)$$

where:  $P$  = load applied to the pyramid, in kg,

$d$  = diagonal of the impression, in mm,

$\alpha$  = angle between opposite faces on the pyramid ( $\alpha = 136^\circ$ ).

**Microhardness measurement.** In certain cases it is necessary to find the hardness of microscopic regions of the metal specimen or even of separate grains of the metal.

Two types of tests, indentation and scratch, are made to determine microhardness.

In *indentation microhardness tests* a diamond pyramid indenter is applied.

Microhardness determination based on scratch tests involves scratching the surface being tested by a diamond under the action of a definite load. The width of the scratch is measured with a special microscope. The hardness value is defined either as the width of the scratch obtained at a certain constant load or as the load at which a scratch of definite width is obtained.

### 6-3. Impact Tests

The impact strength of metals is determined by an impact test on a standard notched specimen.

In the notched-bar impact test, the standard specimen of square or rectangular cross section, notched in the middle, is placed on the supports or anvils of a pendulum-type impact testing machine (Fig. 32a) and is then fractured by a falling pendulum  $Q$  of the weight  $G$ . Pendulum  $Q$ , falling from the height  $h_1$ , fractures the specimen and rises by inertia to the height  $h_2$ . The work done in fracturing the specimen is:

$$A = G (h_1 - h_2) \text{ kg-m.} \quad (58)$$

The impact strength  $a_k$  is the energy required to fracture the specimen referred to unit cross-sectional area of the specimen at the notched cross section.

Impact strength is calculated from the formula:

$$a_k = \frac{A}{F_0} \text{ kg-m per sq cm,} \quad (59)$$

where:  $A$  = energy required to fracture the specimen, in kg-m,  
 $F$  = cross-sectional area of the specimen, in sq cm, at the notch.

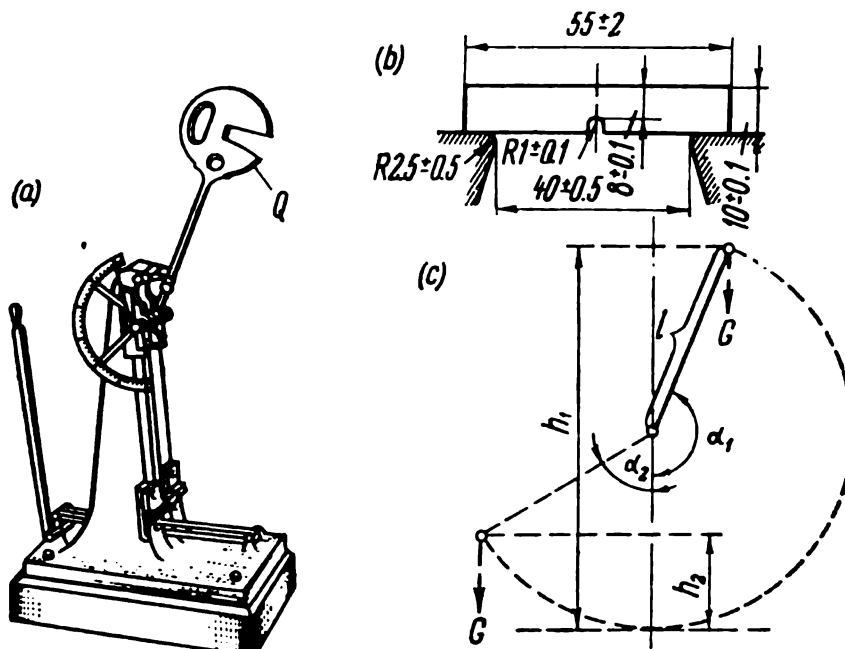


Fig. 32. Notched-bar impact test:  
 (a) impact testing machine; (b) impact test specimen; (c) principle of the impact test

The standard specimen accepted in the U.S.S.R. for impact tests has a square cross section  $10 \times 10$  mm, a length of 55 mm and a U-shaped notch in the middle (Fig. 32b).

## 6-4. Fatigue Tests

The failure of a metal under repeated pulsating or reversing stresses of a value lower than the tensile strength is called *fatigue*. The resistance of a metal to fatigue failure is called the *fatigue*, or *endurance*, *strength* ( $\sigma_w$ ).

Fatigue tests determine the resistance of a metal to repeated pulsating or reversing loads. These tests are made on cylindrical specimens by subjecting them to bending loads during their rotation. This develops reversing stresses (tension-compression) which finally fracture the specimen (Fig. 33a).

A fatigue failure produces a typical fracture in which two zones are quite evident: an external zone, near the surface of the specimen, and an internal zone, in the central part of the specimen (Fig. 33b). The external zone has a smooth surface and is formed as a result of microcracks which, upon development, weaken the cross section of the specimen. The internal zone of the fracture has a rough surface with traces of brittle crystalline failure of the metal. It is formed after the cracks have propagated to occupy a considerable portion of the cross section and have weakened it to a great degree.

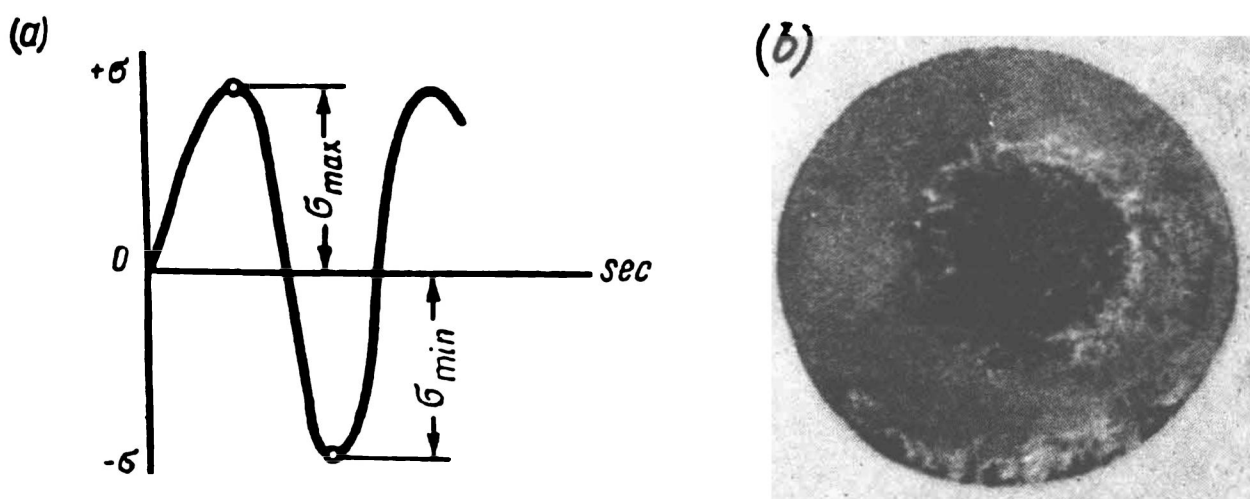


Fig. 33. Fatigue tests:  
(a) cyclic loading; (b) fatigue fracture

Steel specimens which have withstood 5 million load reversals (cycles) without failure indicate that the metal can withstand the same stress for a considerably larger number of cycles.

Specimens of light aluminium alloys are tested on the basis of 20 million cycles.

### 6-5. Fabrication Tests for Metals

The ability of a metal to withstand the deformation it may be subjected to in various types of hot and cold working procedures is determined by fabrication tests on specially prepared specimens.

The results of fabrication tests are usually evaluated by the condition of the surface after the test. The specimen is considered to have passed the test if no external defects such as tears, cracks, lamination or fracture are found on the surface.

The most extensively applied fabrication tests are weld bending, sheet and plate bend, jumping-up and cupping tests.

*Weld bending tests* are conducted to determine the ductility of a butt weld. The specimen rests without clamping on two cylindrical

supports and is bent by a central force until the first crack appears (Fig. 34). The criterion of the ductility is the bending angle  $\alpha$ .

Cold or hot *sheet and plate bend tests* (Fig. 35) are carried out to determine the ability of sheet metal or plate to withstand the effects of bending to a specified size and shape.

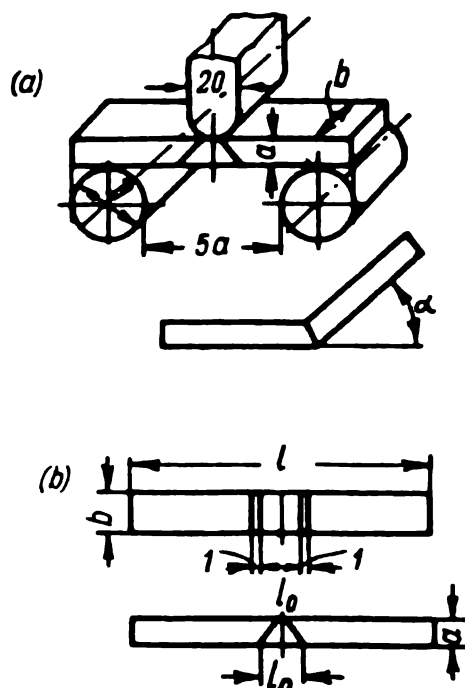


Fig. 34. Weld bending test:  
(a) principle of the test; (b) test specimen

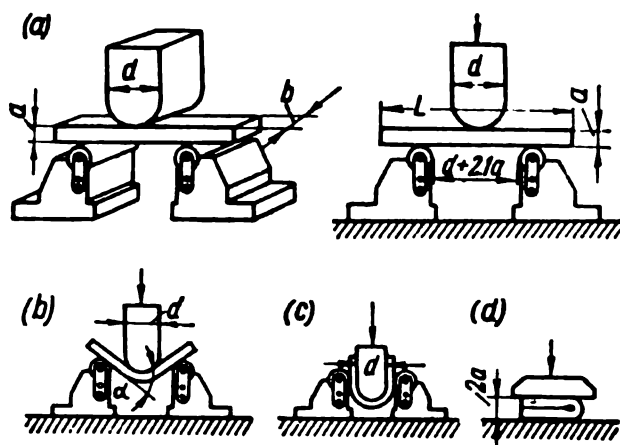


Fig. 35. Plate bend tests:  
(a) principle of the test; (b) bending to a given angle; (c) bending to parallel ends; (d) folding flat

Specimens for these tests are cut from the sheet or plate; the surfaces are not machined. They should have a length of  $L=5a+150$  mm and a width of  $b=2a$ , but not less than 10 mm ( $a$ —thickness of the material).

Such tests are not made on plate over 30 mm in thickness.

Cold *jumping-up tests* determine the ability of metal to undergo compressive deformation of specified size and shape without failure. It is employed for testing metal bar stock used in forging and cold heading operations in the manufacture of bolts, rivets, etc.

*Cupping tests* determine the suitability of sheet metal for cold press-working and drawing, i. e., its formability. A specimen is cut from the sheet or strip and is clamped in the tester (Fig. 36). Then a special punch with a spherical end is forced against the specimen, stretching it into the form of a cup; the test is continued until cracks appear on the cupped part.

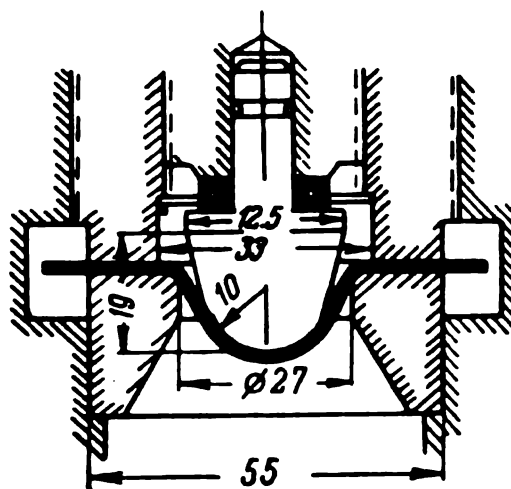


Fig. 36. Principle of the cupping tester



A criterion of the ductility of the metal is the depth to which it is drawn up to the moment of fracture. This cupping depth is stipulated by the specifications according to the material, and the size of the die and punch.

## 6-6. Methods for Investigating the Structure of Metals and Alloys

The mechanical and fabrication properties of metals and alloys depend to a considerable extent on their internal structure which is revealed by macrography, micrography, X-ray analysis and other methods.

**Macrographic investigation of metals.** Macrography involves the study of the structure of metals or alloys as seen by the naked eye or by low-power magnification (up to  $\times 30$ ). The structure of the metal revealed by this investigation is called *the macrostructure*.

A relatively large surface of the metal is studied by macrography. This enables features to be revealed such as the size, form and arrangement of crystalline grains in cast metal; direction of the fibres in deformed metal, shrinkage cavities and blowholes; shrinkage porosity, cracks, chemical inhomogeneity, etc.

The macrostructure is studied directly on the fracture of parts or on special test pieces cut from large ingots, castings or forgings.

**Micrographic investigation of metals.** Micrographic investigations \* on the structure of metals and alloys are carried out with metallographical or electron microscopes.

Metallographical microscopes are extensively used to study the internal structure of metals. They enable the metal to be observed in reflected light with magnifications from  $\times 75$  to  $\times 2500$ . A microscope can be used to determine *the microstructure* of a metal, i. e., the size and shape of its grains (crystallites) and the kind of nonmetallic inclusions it contains.

Micrographic investigations of metals and alloys are conducted on specimens called microsections. These are usually prepared by cutting out small cylinders 10 to 15 mm in diameter and 15 mm high, or 10-mm cubes. One of the flat surfaces of the microsection is smoothed with emery paper and then polished with felt to a mirror finish. This surface either remains unetched or is subsequently etched and then examined under the microscope. Special reagents called etchants (e. g., a 4 to 5 per cent alcoholic solution of nitric acid) are used to etch the microsections to reveal their microstructure (grain boundaries, type of grain structure, etc.).

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\* The microscope was first used to investigate the structure of metals by P. Anosov in 1831.

The electron microscope enables an effective magnification of  $\times 100,000$  to be obtained so that a precise structural and phase analysis of the metal can be made. The high magnification considerably extends the range of application of micrography.

Methods of vacuum metallography, developed in recent years, involve the study of microsections in a vacuum and enable the microstructure of metals to be observed at high temperatures.

Soviet scientists have developed an ultraviolet microscope which can photograph microsections in rays from the ultraviolet region of the spectrum. This feature is very important in interpreting the structure of complex multiple-phase alloys.

## Chapter 7

### THE IRON-CARBON ALLOYS

Absolutely pure iron is very difficult to obtain. In this state it is a soft and very plastic metal of a light-grey colour having a specific weight of 7.86 grammes per cu cm. The mechanical properties of the commercial grades of iron, containing from 0.1 to 0.2 per cent impurities, vary in the following range:

Brinell hardness (Bhn) . . . . .	60 to 80 kg per sq mm
Tensile strength $\sigma_b$ . . . . .	18 to 28 kg per sq mm
Elongation $\delta$ . . . . .	30 to 50 per cent
Reduction in area $\psi$ . . . . .	70 to 80 per cent
Impact strength at 20° C $a_k$ . . . .	16 to 25 kg-m per sq cm

Iron can be alloyed with many elements. Alloys of iron and carbon are the most widely used in engineering; they contain certain amounts of silicon, manganese, chromium, nickel and other elements. Iron-carbon alloys containing up to 2 per cent carbon are called *steels*; those containing more than 2 per cent carbon are called *cast irons*.

### 7-1. The Cooling Curve for Pure Iron

The melting point of pure iron is 1535° C. Iron may exist in several allotropic forms in the solid state, depending upon the temperature to which it is heated. The transformation of iron from one allotropic form to another is revealed by temperature effects (retardations) on the cooling or heating curves. Fig. 37 illustrates the cooling curve for pure molten iron plotted in time vs temperature coordinates. The first horizontal step appears on this curve at a temperature of 1535° C.

It indicates that heat is evolved and that the iron passes from the liquid to the solid state. Upon freezing the melt delta iron is formed having a body-centred cubic crystal lattice with a constant  $a=2.93$  Å. The second temperature effect (point  $A_2$ ) occurs at  $1390^\circ\text{C}$  on the iron cooling curve and corresponds to the transformation of delta iron into a new allotropic form—gamma iron. Gamma iron has a face-

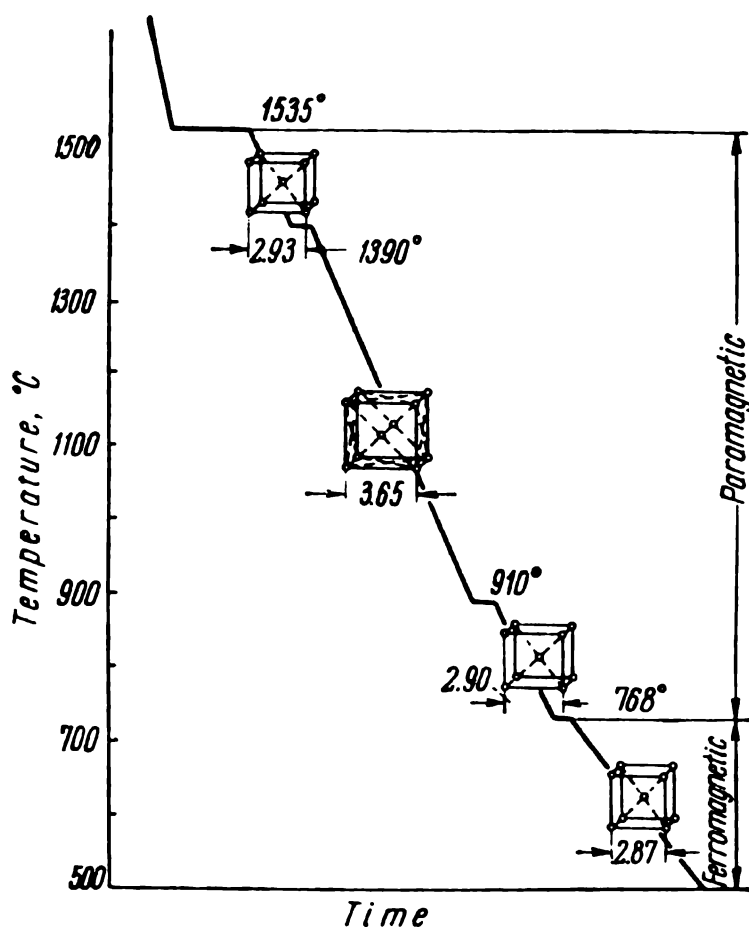


Fig. 37. The cooling curve for pure iron

centred cubic lattice with a constant  $a=3.65$  Å. Next on the cooling curve is the third step (point  $A_3$ ) at  $910^\circ\text{C}$ . Here, gamma iron is transformed into beta iron with a body-centred cubic lattice with a constant  $a=2.90$  Å. The last, fourth, retardation is observed at  $768^\circ\text{C}$  (point  $A_4$ ). It corresponds to the transformation of beta iron into alpha iron with the same type of crystal lattice as beta iron but with a constant  $a=2.87$  Å. Heat is evolved in the beta to alpha iron transformation, not because of a rearrangement of the space lattice but due to intra-atomic changes as a result of which alpha iron acquires pronounced ferromagnetic properties.

Previously it was thought that the appearance of ferromagnetic properties in the cooling of iron is the result of an allotropic transformation and therefore paramagnetic iron in the interval from  $768^\circ$

to  $910^{\circ}\text{C}$  was called beta iron. Later it was established that the space lattice does not change in the alpha-to-beta transformation. Consequently, beta iron must be regarded as a paramagnetic state of alpha iron.

Delta iron, whose existence is observed in the interval from  $1535^{\circ}$  to  $1390^{\circ}\text{C}$ , has a space lattice of the body-centred cubic type, i. e., the same as alpha iron. Thus, there are actually two allotropic forms of iron: the low-temperature modification, alpha iron, and the high-temperature form—gamma iron.

## 7-2. The Iron-carbon Equilibrium Diagram

The binary iron-carbon system is the basis of the engineering alloys of iron (steel and cast iron).

The iron-carbon equilibrium diagram in alloy composition vs temperature coordinates is given in Fig. 38a. It was developed by the work of a number of investigators beginning with D. Chernov who discovered the critical points in steel in 1868. Pure iron is on the left side of the diagram and cementite ( $\text{Fe}_3\text{C}$ ), containing 6.67% C, on the right side. *Cementite* is a chemical compound of iron and carbon and may form upon rapid cooling of the iron-carbon melt from high temperatures. The structural constituents of the freezing alloy, in this case, are not sufficiently stable (they are metastable) since they do not reach phase equilibrium. If the iron-carbon alloy is slowly cooled, free graphite, i. e., chemically pure carbon, may separate from the melt. The resulting structural constituents are more stable as they come to phase equilibrium.

Thus, there are two versions of the iron-carbon equilibrium diagram:

- (1) metastable diagram for the iron-cementite system (solid lines),
- (2) stable diagram for the iron-graphite system (dash lines).

We shall first consider the metastable version of the iron-carbon equilibrium diagram (solid lines) in which the structures formed in the solidified phases do not reach sufficiently complete equilibrium.

**Upper part of the diagram.** Solidification of the liquid iron-carbon melt begins on line *ABCD* (liquidus); above the liquidus (area *I*) the alloy is in a liquid state and is a homogeneous system. Crystals of the solid solution of carbon in delta iron separate from the melt along the liquidus *AB*. Their composition is indicated by line *AH*. Point *H* represents the maximum solubility of carbon (0.1 per cent) in delta iron at  $1490^{\circ}\text{C}$ . A chemical reaction proceeds along line *HJB* at the constant temperature of  $1490^{\circ}\text{C}$  between the crystals of delta iron saturated with carbon (point *H*) and the liquid containing 0.5% carbon (point *B*). The reaction results in the formation of a new substance—austenite, which is the solid solution of carbon in gamma iron.



*Austenite* has a face-centred cubic space lattice. The carbon content of the austenite formed along line *HJB* is 0.18 per cent (point *J*). The maximum solubility of carbon in austenite is 2 per cent at 1130° C (point *E*). Austenite is paramagnetic, soft and its Brinell hardness is Bhn=170-200.

Crystals of austenite of variable chemical composition with the carbon content ranging from 0.18 to 2 per cent separate from the melt along the line *BC*.

The complete solidification of these alloys proceeds along line *HJEC* (solidus). Alloys containing from 0.18 to 2 per cent carbon become solid along the part of the solidus *HJE*. Those with from 2 to 4.3 per cent carbon are completely solid on line *EC*. The solidification of the last portions of the liquid phase, enriched in carbon to 4.3 per cent, takes place along this line. All the liquid is completely solidified at 1130° C; at the same time crystals of austenite, containing 2% C (point *E*), and cementite, containing 6.67% C (point *F*), separate from it. These crystals of austenite and cementite form the eutectic mixture known as *ledeburite* (point *C*). Thus, below line *EC* the solidified alloy consists of austenite and ledeburite.

The eutectic alloy at point *C*, containing 4.3% C, solidifies at the constant temperature of 1130° C with the formation of only ledeburite.

Ledeburite is distinguished by its high hardness (Bhn=700) and brittleness. In the temperature interval 1130°-723° C ledeburite is a mechanical mixture of grains or plates of austenite and cementite; below 723° C it consists of grains or plates of cementite and products of the further decomposition of austenite.

The solidification of alloys containing from 4.3 to 6.67% C begins along line *CD* with the separation of primary cementite ( $\text{Fe}_3\text{C}$ ) from the melt. Cementite is very hard and brittle; its Brinell hardness is about 800. It is weakly magnetic and is a poor conductor of electric current and heat.

In the course of the separation of cementite containing 6.67% C from the liquid phase, the latter becomes leaner in carbon. Its amount is also reduced. When line *CF* is reached, the remaining melt attains the eutectic composition (4.3% C) and freezes at the constant temperature of 1130° C. The resulting solidified alloy consists of primary cementite and ledeburite below line *CF*.

Between the liquidus *ABCD* and the solidus *AHJECF* (areas *II* and *III*) iron-carbon alloys comprise two phases—liquid and solid, the amount of solid phase increasing with a fall in temperature. In alloys containing up to 4.3% C (hypoeutectic alloys) the composition of the solid phase varies along line *AHJE* and that of the liquid phase—along line *ABC*. Alloys of this type will consist of austenite crystals and the liquid phase in the area between the liquidus and solidus (*II*).

In alloys containing from 4.3 to 6.67% C (hypereutectic alloys) the composition of the liquid phase varies along line *DC* with a reduction of temperature while that of the solid phase corresponds to the composition of cementite. Between the liquidus and solidus these alloys consist of primary cementite crystals and the liquid phase (area *III*).

Further changes in structure are observed in the solidified alloys at temperatures below the solidus *AHJECF*. These changes are associated with what is called secondary crystallisation or recrystallisation in the solid state.

**Lower part of the diagram.** We shall next consider the transformations that occur in the solidified alloys below the solidus *AHJECF*. They are caused by the following circumstances:

- (1) as the temperature falls, iron changes from one allotropic form to another (gamma iron  $\rightarrow$  alpha iron),
- (2) the solubility of carbon in gamma and alpha iron is substantially reduced as the temperature decreases.

Hypoeutectic iron-carbon alloys with a carbon content up to 2 per cent, in the area below the solidus *AHJE* but above line *GOSE*, consist solely of austenite.

In cooling alloys containing less than 0.8% C below line *GOS* the austenite decomposes with the separation of ferrite.

*Ferrite* is an interstitial solid solution of carbon in alpha iron. At room temperature ferrite contains 0.006% C. The maximum carbon content of ferrite is 0.05 per cent at 723° C (point *P*). In addition to carbon, a certain amount of silicon, manganese and phosphorus may be dissolved in the ferrite. The ferrite space lattice is of the body-centred cubic type, the same as for pure alpha iron. Since the ferrite separating upon cooling austenite below line *GOS* contains not over 0.05% C, the remaining austenite becomes richer in carbon. Its composition varies along line *GOS* and finally reaches point *S* at 723° C.

In point *S* at 723° C and at a carbon content of 0.8 per cent, austenite decomposes with the simultaneous separation of both ferrite and cementite which form the eutectoid mixture called *pearlite*.

Under a microscope pearlite is seen as an intimate mixture of very thin plates (lamellae) or fine grains of cementite and ferrite; pearlite may be either lamellar or granular. In lamellar pearlite the cementite is distributed in the ferrite in the form of plates; in granular pearlite cementite is in the form of rounded globules distributed as separate grains in the ferrite matrix.

The mechanical properties of pearlite depend upon the fineness of the cementite particles ( $\sigma_b$  = up to 82 kg per sq mm,  $\delta$  = up to 15 per cent and Bhn = up to 160).

The alloy containing 0.8% C (point *S*) is called *the eutectoid alloy*. Upon cooling the eutectoid alloy below 723° C, all of the austenite

is transformed into pearlite. Alloys to the left of point *S* are called *hypoeutectoid steels*; those to the right and up to 2% C are called *hypereutectoid steels*.

When hypereutectoid alloys are cooled below line *SE*, the austenite decomposes with the separation of cementite. The latter is called *secondary*, or *proeutectoid*, cementite to distinguish it from the primary cementite that separates from the melt along line *CD*. Since this proeutectoid cementite contains 6.67% C, the carbon concentration in the remaining austenite changes continuously along line *SE* up to the composition at point *S* (0.8% C) at 723° C. Below this line the remaining austenite is transformed into pearlite.

Thus, only austenite is found in hypoeutectoid steels (up to 0.8% C) at temperature above line *GOS*, i. e., in area *IV*. Below line *GOS* but above line *PSK* (area *VIII*) the alloy comprises austenite and ferrite. In hypereutectoid steels (from 0.8 to 2% C), the structure consists of austenite and proeutectoid cementite between line *ES*, determining the solubility of carbon in gamma iron, and line *PSK*, i. e., area *V*. Below line *PSK* alloys containing up to 0.8% C consist of pearlite and ferrite (area *IX*) and alloys with from 0.8 to 2% C (area *X*)—of proeutectoid cementite and pearlite. Alloys having a carbon content from 2 to 4 per cent consist of austenite, proeutectoid cementite and ledeburite below line *EC* but above line *PSK* (area *VI*).

When these alloys are cooled below line *PSK* (area *XI*) the grains of austenite are transformed into pearlite so that the new structure comprises pearlite, proeutectoid cementite and ledeburite. Alloys of the eutectic composition (point *C*) contain ledeburite alone below line *PSK*. Alloys with a composition of from 4.3 to 6.67% C consist of primary cementite and ledeburite between lines *CF* and *PSK*, i. e., in area *VII*. This same structure is retained below line *PSK*, in area *XII*.

The line *PQ* in the lower left part of the diagram indicates the reduction in solubility of carbon in alpha iron (ferrite) with a decrease in temperature. Tertiary cementite is separated from the ferrite along line *PQ* and thus all alloys between points *P* and *Q* consist of ferrite and tertiary cementite. Alloys in the area *QPG* consist of ferrite alone.

Fig. 38*b* illustrates a simplified iron-carbon equilibrium diagram showing the alloy structure in the various areas of the diagram.

### 7-3. Microstructure of Iron-carbon Alloys

The iron-carbon equilibrium diagram enables the internal structure of steels and cast irons to be tied in with their chemical composition and cooling conditions. The microstructures obtained when steel specimens of various carbon content are cooled slowly are illustrated



in Fig. 39. Steel of very low carbon content (0.02 per cent) consists of ferrite alone (Fig. 39a) which is observed under the microscope as light-coloured grains of varied size and shape on the etched surface of the microsection. The size and shape of the ferrite grains depend upon the kind and degree of deformation in rolling or forging.

As the carbon content is increased, a new structural constituent of iron-carbon alloys, pearlite, is found in the photomicrographs of hypoeutectoid steels. In low-carbon steels pearlite is seen as separate small dark inclusions between the light-coloured grains of ferrite (Fig. 39b). An increase in the carbon concentration in the steel leads to a proportional increase in the amount of pearlite (Fig. 39 c, d and e). At a carbon content of 0.80 per cent pearlite is distributed over the whole field of view in the microscope (Fig. 39f). As a rule, pearlite has either a lamellar or a granular structure (Fig. 39g). In the first case at low magnifications of the microscope (up to  $\times 100$ ) the pearlite seems to consist of light and dark stripes. In the second case the pearlitic structure is characterised by a uniform distribution of the cementite in the ferrite in the form of round globules or dots.

Lamellar pearlite may be transformed into the granular modification by holding it at about  $700^{\circ}\text{C}$  for a considerable period of time. In hypereutectoid steels containing more than 0.80% C a new phase is seen on the field of the pearlite structure. This is proeutectoid cementite which usually has the form of a light-coloured network between the pearlite grains (Fig. 39h).

The amount of cementite in hypereutectoid steel increases continuously with an increase in the carbon content. At a content of 2% C, cementite occupies 18 per cent of the microscope field of view. Even at this maximum carbon content for hypereutectoid steels, the proeutectoid cementite retains the form of a network and is not precipitated as separate grains.

The microstructure of iron-carbon alloys containing over 2% C, i. e., cast irons, depends not only upon their chemical composition but also upon the cooling rate in solidification of the liquid phase.

In rapidly cooled molten cast iron the carbon may separate in the form of cementite disseminations embedded in the metal matrix.

In white cast iron all of the carbon is in the form of cementite, i. e., in the combined state.

White cast irons are very brittle and hard. They cannot be machined and are usually employed in manufacturing steel or malleable cast iron. Fig. 40 illustrates the main types of microstructure of white cast iron with various carbon content. The white structural constituent, cementite, can be seen distinctly on all the photomicrographs; its amount is proportional to carbon concentration in the cast iron.

In the hypoeutectic cast irons (Fig. 40 a and b) the cementite is located between the darker grains having the form of the austenite

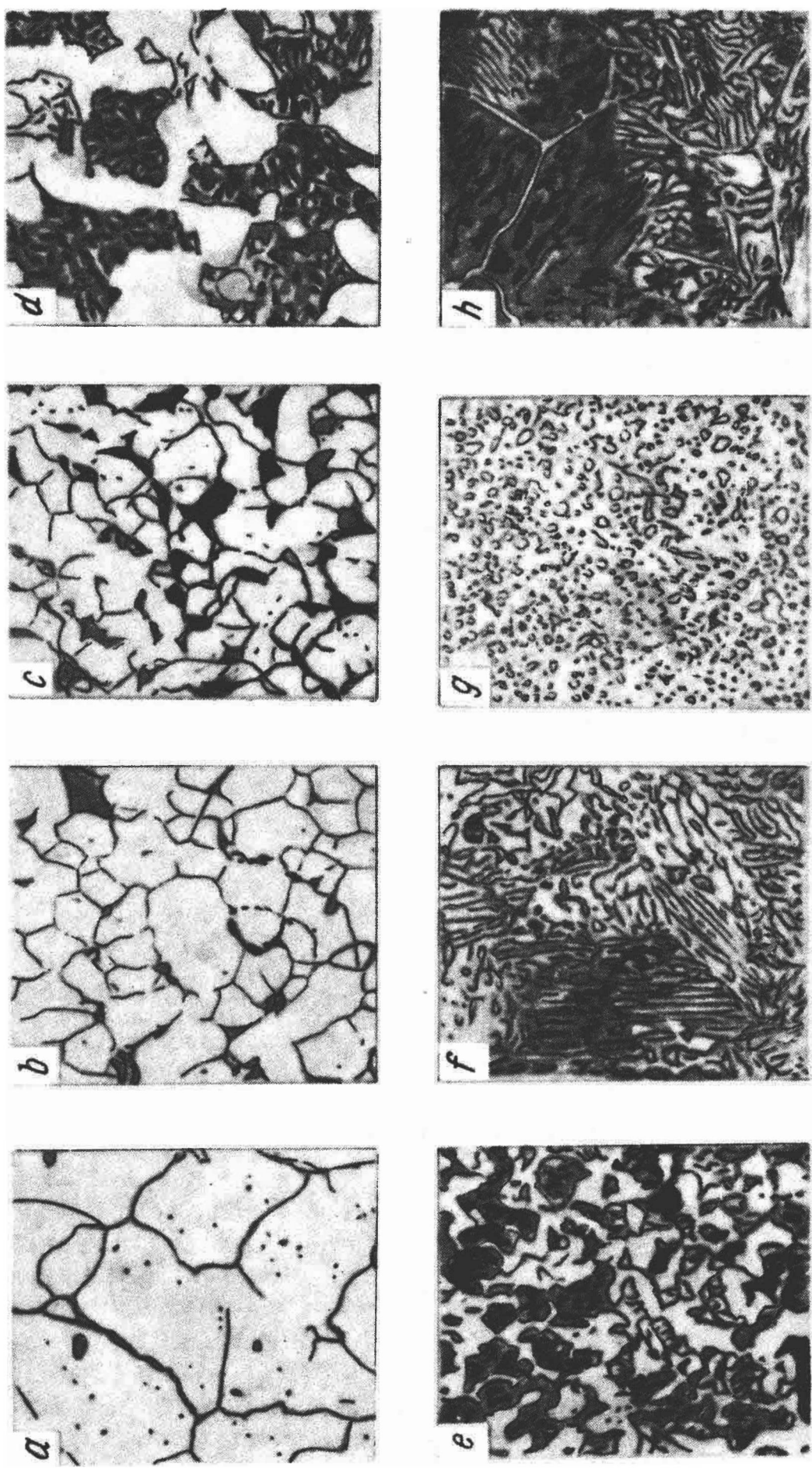


Fig. 39. Microstructures of steels of various carbon contents

that separated during the freezing of the liquid metal. Since austenite can only exist at high temperatures (above line *PSK*,  $723^{\circ}\text{C}$ ) the dark grains observed in all specimens of white cast iron under the microscope at room temperature are not austenite but the products

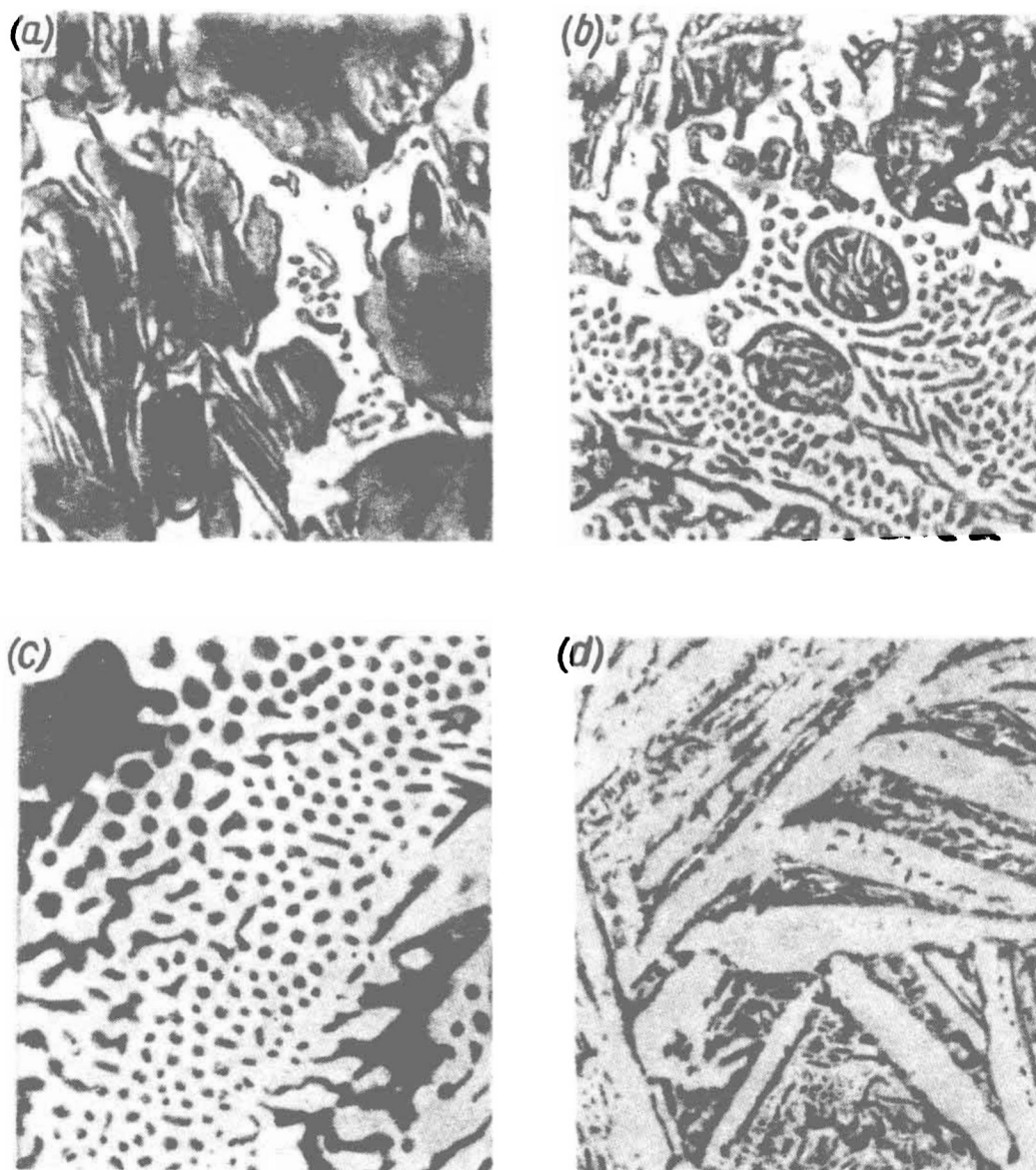


Fig. 40. Microstructures of white cast irons

of its decomposition, i. e., pearlite (intimate mixture of ferrite and cementite). These austenite decomposition products, however, are located in the solidified white cast iron exactly in the same places where the austenite initially was. Their form also corresponds to that of the initially separated austenite. Consequently, the microstructure of white cast irons retains the appearance that it had at high temperatures (between lines *ECF* and *PSK*) when the cast iron consisted of undecomposed solid solution of carbon in gamma iron (austenite) and cementite.

The most interesting microstructure is the one observed in white cast iron of the eutectic composition having 4.3% C (Fig. 40c). The whole field of view of the microscope is occupied by ledeburite in which the dark inclusions are the products of austenite decomposition (pearlite) and the light constituent is cementite. Ledeburite, the eutectic mixture of austenite and primary cementite, forms at 1130° C when the molten cast iron solidifies. Austenite decomposition takes place much later, when the cast iron is cooled below line *ECF*. It should be noted that the ledeburite is retained in hypoeutectic cast irons as well, where it precipitates along the austenite grain boundaries.

Ledeburite is also found in the structure of hypereutectic cast irons but here it is located between the light needles of primary cementite which separated from the melt during freezing (Fig. 40d).

If molten cast iron is slowly cooled, the carbon dissolved in it may separate from the melt in the form of graphite. Graphite is distributed within the metal matrix as independent inclusions which impart the greyish colour to the fracture of cast iron. It is for this reason that cast irons in which free graphite is embedded are called *grey cast irons*. The precipitated graphite of grey cast irons may have the shape of the thinnest streaks or plates (lamellae), spheroidal nodules or flakes. Graphite weakens the metallic matrix of grey cast iron making it susceptible to impact loads.

The mechanical properties of grey cast iron can be substantially improved by controlling the size of the graphite inclusions, and by obtaining finer inclusions more uniformly distributed in the matrix.

The process of graphitisation is controlled by varying the rate of cooling and by proper alloying of the metallic matrix. It has been established that various elements added to the cast iron in smelting have different effects on the matrix and on the degree of graphitisation of the cast iron. For example, silicon promotes carbon precipitation and improves the castability by making the cast iron somewhat softer. The silicon content of grey cast irons usually ranges from 0.8 to 4.5 per cent. Manganese, on the contrary, retards graphitisation and promotes the formation of combined carbon, i. e., it tends to "chill" the cast iron. A manganese content in cast iron up to 1.2 per cent is desirable since it increases the strength and hardness, and eliminates the harmful effects of sulphur by combining it in the form of manganese sulphide (MnS) (insoluble in metal) which goes into the slag. At a higher manganese content the cast iron becomes brittle. Phosphorus increases the fluidity of the molten cast iron and, therefore, its presence in small amounts (up to 0.4 per cent) is permissible. The phosphorus content should be as low as possible in critical grey iron castings (not more than 0.15 per cent) since it increases the brittleness of the metal.

Sulphur is an especially harmful impurity; it impedes the precipitation of graphite and increases the shrinkage and brittleness of cast iron, making it less fluid. No more than 0.1% S may be allowed in cast irons.

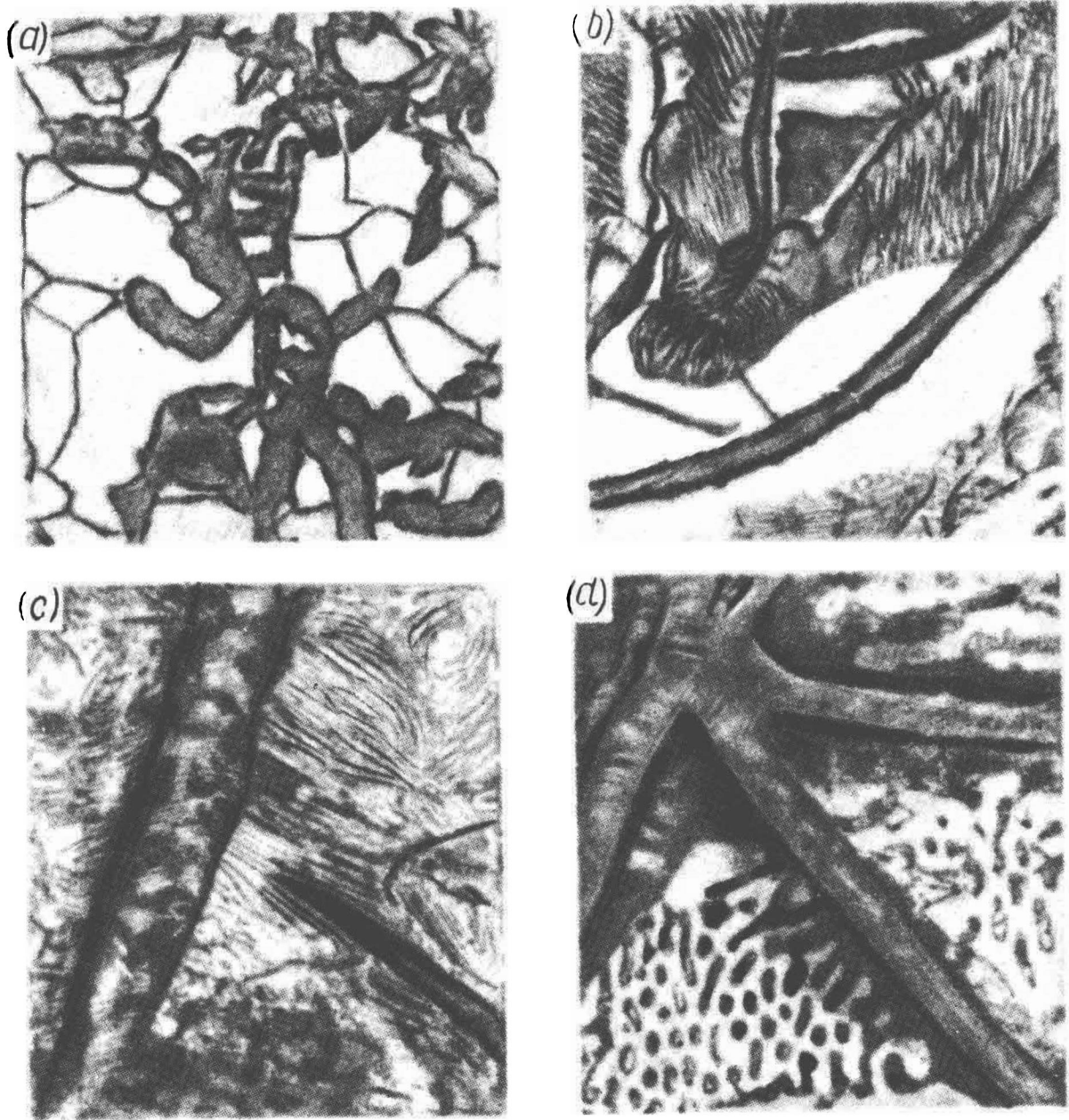


Fig. 41. Microstructures of grey cast irons

The microstructures of grey cast irons of various chemical compositions are shown in Fig. 41.

According to the form of the carbon present and the nature of the metallic matrix, grey cast irons are classified as:

(1) *ferritic*, in which all the carbon is in the free state as graphite. Such cast irons have a structure consisting of ferrite (light-coloured grains) and graphite (dark flakes) (Fig. 41a);

(2) *ferrito-pearlitic*, in which part of the carbon is in the combined form (as  $\text{Fe}_3\text{C}$ ), being the component of pearlite (up to 0.8% C may



go into the pearlite). These cast irons have a structure of lamellar pearlite+ferrite (light-coloured grains)+graphite (black flakes) (Fig. 41b);

(3) *pearlitic*, containing 0.8% C in the form of  $\text{Fe}_3\text{C}$ . Their structure is lamellar pearlite+graphite (black flakes) (Fig. 41c);

(4) *mottled* (pearlito-cementitic), in which the outer layers have the structure of white cast iron and the core, that of grey cast iron (Fig. 41d). Its structure is pearlite (granular or lamellar)+cementite (light-coloured needles and veins) + graphite (black flakes).

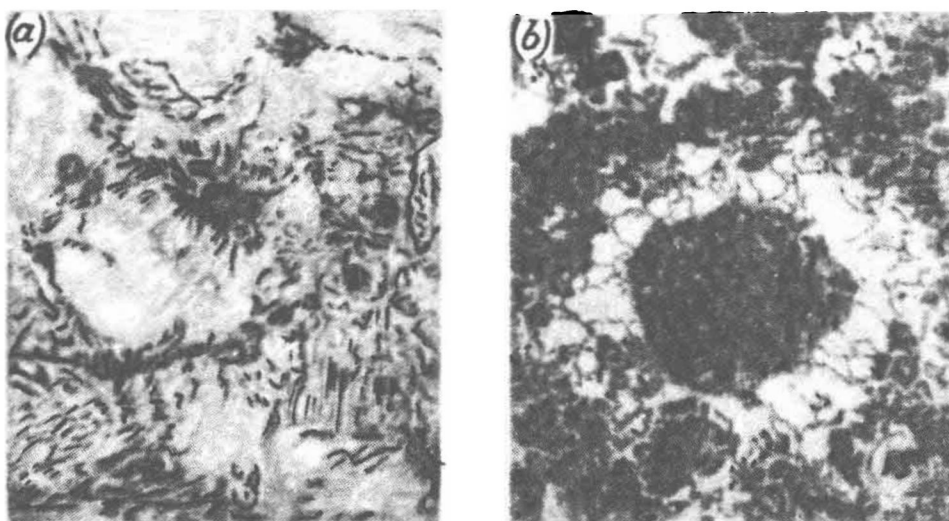


Fig. 42. Microstructures of high-strength cast iron

Grey cast irons with a metallic matrix of pearlitic structure and uniformly distributed fine graphite flakes have the highest mechanical strength and are known as *high-quality cast irons*.

Recently, methods have been developed for obtaining high-strength cast iron with a tensile strength up to 70 kg per sq mm, elongation up to 10 per cent and a Brinell hardness up to 269.

High-quality cast irons are obtained by making very small additions to liquid cast iron of a definite composition in the tapping spout or ladle when pouring from the cupola. These additions are called *inoculants*; they control the structure of the metallic matrix in the cast iron (make it pearlitic) and increase the strength of the metal without appreciably changing its chemical composition.

Magnesium or ferrosilicon (an iron-silicon alloy) is most frequently used as an inoculant in foundry practice. It is added in amounts from 0.1 to 0.6 per cent by weight.

Fig. 42a illustrates a high-strength cast iron before inoculation; the ferrite-pearlitic structure of the metal is quite evident. The same cast iron is shown in Fig. 42b after inoculating with magnesium or

ferrosilicon. Spheroidal graphite has been formed and the cementite has decomposed.

Another method of graphitisation should be mentioned here. If white cast iron is subjected to prolonged annealing (malleablisation) at  $900-1000^{\circ}\text{C}$ , i. e., considerably above line *PSK* but below line *ECF*, the combined carbon will be transformed into *temper carbon*. This process changes the structure of white cast iron and its mechanical properties. What is obtained is called *malleable cast iron*.

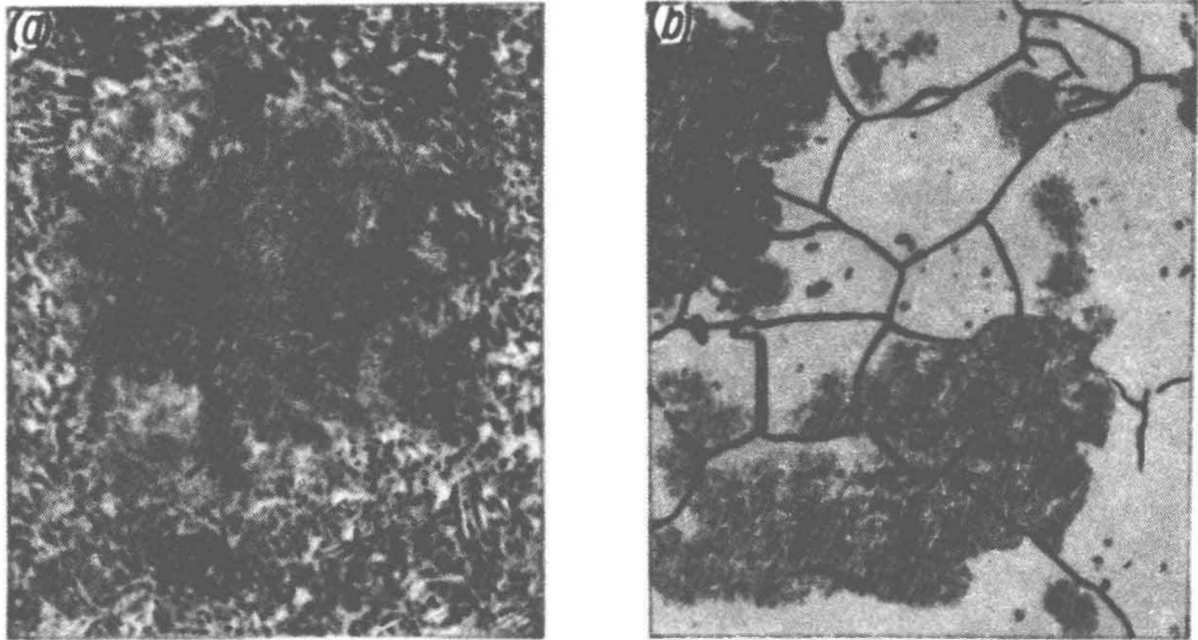


Fig. 43. Microstructures of malleable cast iron

Malleable cast iron differs from grey cast iron in that the graphite is precipitated in the form of separate grains or as rounded clusters which are isolated from each other in the metallic matrix. This matrix in malleable cast iron is similar in structure to that of grey cast iron in the sense that it may be either ferritic or pearlitic. Hence, malleable cast irons are classified into two types:

- (1) pearlitic, with a structure of pearlite and temper carbon (Fig. 43a); the pearlite may be either lamellar or granular;
- (2) ferritic, with a structure of ferrite and temper carbon (Fig. 43b).

#### 7-4. Transformations that Occur in Heating and Cooling Steel

**Transformations in steel during heating.** The structure of iron-carbon alloys at room temperatures depends upon their chemical composition and the cooling rate below lines *GOSECF* and *PSK* in the iron-carbon equilibrium diagram (see Fig. 38).

As a result of slow cooling, hypoeutectoid steels acquire a structure of ferrite and pearlite, eutectoid steels—pearlite alone and hy-

pereutectoid steels—pearlite and proeutectoid cementite. If any one of these steels is heated it will undergo structural transformations.

The temperatures at which structural transformations occur in steel in the solid state are called *critical points* and are designated by the letter *A* which is followed by the subscript *c* when referring

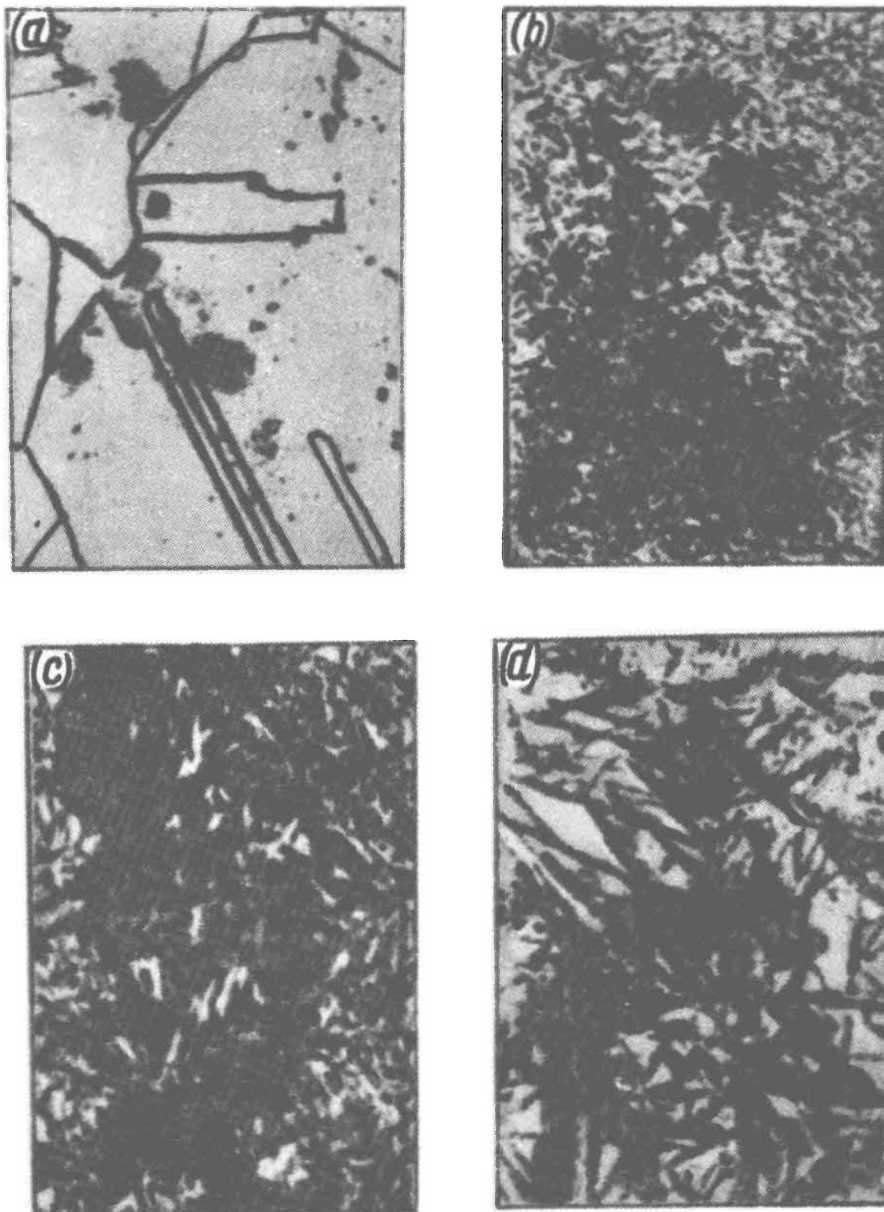


Fig. 44. Structural constituents of steel:  
(a) austenite; (b) sorbite; (c) troostite; (d) martensite

to heating and  $r$  in cooling. The critical points representing the transformations in steels on line *PSK* are denoted by  $A_c$ , and  $A_r$ , those on line *MO* by  $A_{c_1}$  and  $A_{r_1}$ , those on line *GOS* by  $A_{c_2}$  and  $A_{r_2}$ , and those on line *SE* by  $A_{cm}$ .

We shall now consider the transformations that may occur in heating steel from room temperature to temperatures above the critical points.



It follows from the iron-carbon equilibrium diagram that when a hypoeutectoid steel is heated to a temperature between  $A_{c1}$  and  $A_c$ , the pearlite begins to transform into austenite and the steel consists of austenite and ferrite. Upon further heating of this steel above the upper critical point  $A_c$ , all of the ferrite is dissolved in the austenite and the structure becomes purely austenitic (Fig. 44a).

Eutectoid steel containing 0.8% C consists of pearlite until it is heated to a temperature of 723°C. At point  $A_c$ , (723°C) all of the pearlite is transformed into austenite.

Up to a temperature of 723°C (point  $A_{c1}$ ), hypereutectoid steels, containing from 0.8 to 2% C, have a structure of pearlite and proeutectoid cementite. As these steels are heated above  $A_{c1}$ , the pearlite begins to transform into austenite and the structure of hypereutectoid steels between points  $A_{c1}$  and  $A_{cm}$  consists of austenite and proeutectoid cementite.

At temperatures above  $A_{cm}$  the last of the surplus cementite dissolves in the austenite and the hypereutectoid steels acquire a purely austenitic structure. Hypereutectoid steels are never heated above point  $A_{cm}$  in heat treatment, i. e., above line  $SE$ , since cementite has a high hardness and it is expedient to retain it in the metal structure. Hence, to obtain austenite in the structure of hypereutectoid steels with the aim of subsequently fixing this structure, the steel is heated to a temperature 20° to 30° above point  $A_{c1}$ , i. e., 20° to 30° above line  $PSK$ . Hypoeutectoid steels are heated for the same purpose 20° to 30° above point  $A_c$ , i. e., 20° to 30° above line  $GOS$ .

**Transformations in steel during cooling.** We shall next consider the transformations that occur in steels of various compositions when they are cooled at different rates.

If austenite is obtained in a steel and then the heated metal is slowly cooled, the austenite will decompose below line  $PSK$  to form the mechanical mixture of ferrite and cementite (pearlite).

An increased rate of cooling of the steel, corresponding to an increase in the degree of supercooling of the austenite in reference to line  $PSK$ , leads to the formation of a finely dispersed pearlite known as *sorbite* (Fig. 44b).

If the steel is cooled at a still higher rate, an even finer structure of austenite decomposition is obtained, which is called *troostite* (Fig. 44c).

Since the cementite in sorbite and troostite is in a finely divided (dispersed) state, the latter two constituents are harder than pearlite. While pearlite has a Brinell hardness of 180, that of sorbite is 350 and troostite—450.

Though troostite and sorbite are also mixtures of ferrite and cementite they differ from pearlite in that they have a variable chemical

composition and a more finely dispersed structure. A finely dispersed two-phase structure can still be distinguished in sorbite under a microscope at magnifications over  $\times 1000$ . Troostite, however, has the appearance of an uninterrupted dark-coloured mass due to its high susceptibility to etching. The high hardness of sorbite and troostite is a result of their finely dispersed structure.

Steel can be cooled from the high-temperature region at a rate so high that the austenite does not have sufficient time to decompose into troostite or sorbite. In this case the austenite is transformed into a new structural constituent of acicular (needlelike) form called *martensite* (Fig. 44d).

Martensite is a supersaturated solid solution of carbon in alpha iron. The carbon content of martensite is the same as that of the austenite from which it is formed. Martensite is ferromagnetic, very hard and brittle. Its Brinell hardness ranges from 650 to 750. Such a high hardness of martensite is explained by the distortion of the alpha-iron crystal lattice by interstitial carbon atoms.

The formation of martensite is accompanied by an increase in the volume of the steel by about 3 per cent. This results in internal stresses produced by rapid cooling (quenching).

Thus, by controlling the cooling rate of steel, or the corresponding degree of supercooling of the austenite, a series of intermediate structures may be obtained between austenite and its final decomposition product—pearlite.

The effect of the cooling rate of eutectoid carbon steel upon the position of its critical point  $A_{r_1}$  is shown in Fig. 45.

At very slow cooling, austenite decomposes near the temperature of  $723^\circ\text{C}$  with the formation of pearlite at point  $A_{r_1}$ .

An increased cooling rate leads to supercooling of the austenite and, as a result, to its decomposition in a certain temperature interval with the formation of the intermediate structures, sorbite and troostite. The critical point  $A_{r_1}$  continuously falls at more rapid cooling and reaches the position of point  $A_{r'_1}$  on the diagram. At a cooling rate of  $v_1$ , a second point  $A_{r''_1}$ , or  $M_s$ , appears on the diagram. It indicates the temperature at which austenite starts to transform into martensite. Therefore, when steel is cooled at the rate  $v_1$ , martensite appears in addition to troostite, though a certain amount of austenite is retained. The points  $A_{r'_1}$  and  $A_{r''_1}$  are observed at cooling rates up

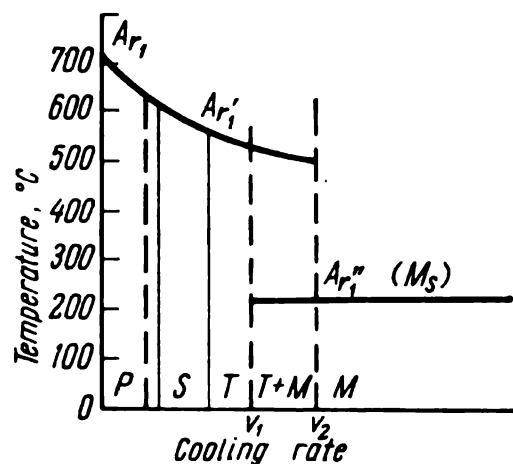


Fig. 45. Effect of the cooling rate of eutectoid steel on the position of critical point  $A_{r_1}$ .

to  $v_2$ ; at a further increase in the cooling rate, point  $A_{r'}$  disappears and austenite is transformed only into martensite.

The minimum cooling rate  $v_c$  at which the austenite is transformed into martensite alone is called *critical cooling rate in quenching steel*.

### 7-5. Isothermal Decomposition of Austenite

Austenite transformation, or decomposition, which is the same thing, may proceed either upon continuous cooling of steel considerably below point  $A_{r'}$  or when it is held at a constant temperature slightly below point  $A_{r'}$  (isothermal transformation of austenite).

The most comprehensive conception of the kinetics involved in the structural changes of the solid solution of carbon in gamma iron may be gained by studying the isothermal decomposition of supercooled austenite, i. e., its transformation at constant temperature.

The isothermal transformation of austenite has been investigated experimentally in great detail. This was done by heating steel specimens above the critical points  $A_{c_1}$ , then rapidly cooling them to 700°, 600°, and 400° C to obtain austenite and, finally, holding the specimens at these temperatures for various lengths of time.

The transformation of austenite at constant temperatures does not begin immediately, but only after a certain time interval which is called the incubation period. The isothermal austenite decomposition diagram for eutectoid steel (0.8% C) is shown in Fig. 46. These are usually called *TTT* diagrams as they relate the *transformation* of the austenite with the *time* and *temperature* conditions to which it is subjected. Other names for these diagrams are *S-curves* and *C-curves*. In such diagrams time is plotted in seconds along the abscissas in a logarithmic scale and the temperatures in degrees Centigrade with an ordinary scale, along the ordinates.

Fig. 46 shows that at 700° C the steel retains its austenitic structure for a certain period of time. Austenite decomposition starts at point  $a_s$  and, after a certain time, finishes at point  $a_f$ . At a temperature of 600° C the austenite transformation begins at point  $b_s$  and ends at point  $b_f$ . The same points for the process at 400° C are  $c_s$  and  $c_f$ . The curves 1 and 2 are then drawn through all the points indicating the beginning and end of austenite decomposition. Curve 1 represents the conditions at which austenite begins to decompose while curve 2 shows those at which austenite decomposition is completed. Austenite is stable above the dash line  $A_{r'}$ . The region to the left of curve 1 corresponds to supercooled austenite which is denoted by  $A'$ ; that to the right of curve 2—to the complete transformation of austenite. The interval between the two curves indicates conditions for the partial decomposition of austenite into ferrite and cementite.

The point at which austenite begins to transform into martensite is denoted by  $M_s$  (see Fig. 46). In eutectoid steel the martensite transformation begins at  $240^\circ\text{C}$  and ends at  $50^\circ\text{C}$  below zero. This last point is denoted by  $M_f$ . The positions of the  $M_s$  and  $M_f$  points depend upon the chemical composition of the steel; they are not affected by the cooling rate.

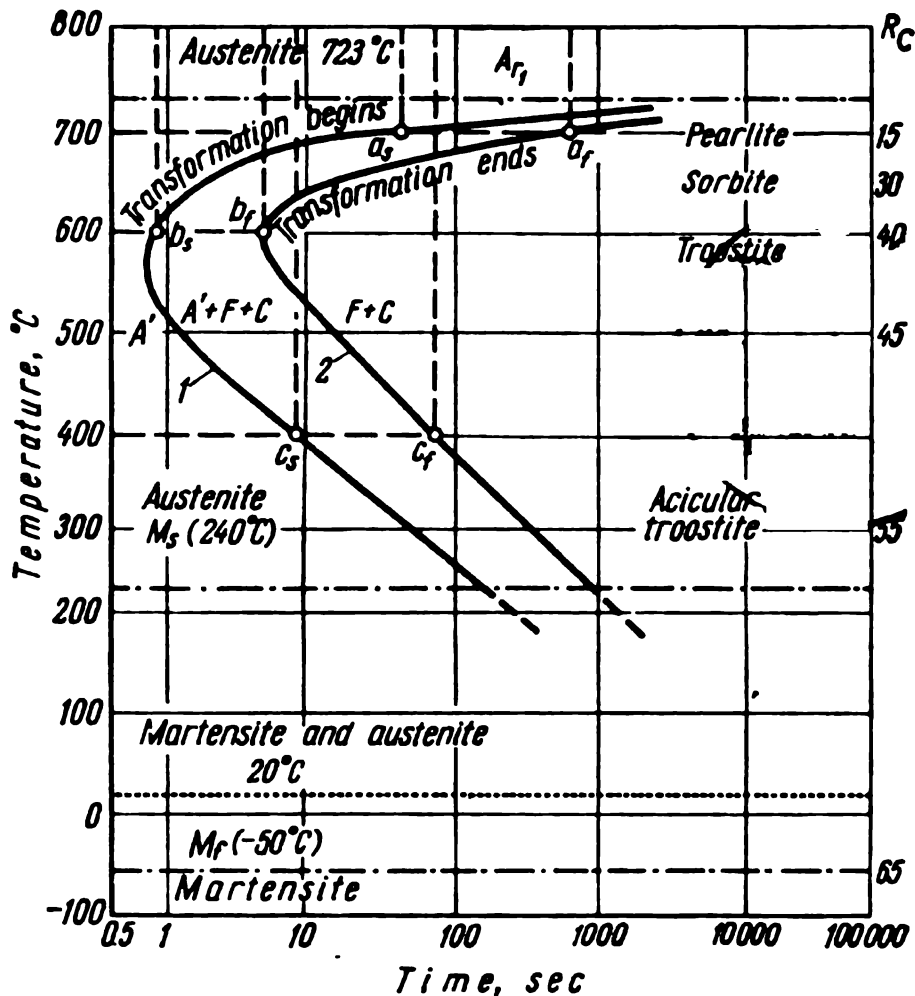


Fig. 46. Isothermal austenite decomposition (TTT) diagram for carbon eutectoid steel

For the steel being considered, supercooled austenite is least stable at  $550^\circ\text{C}$  and it can be held without decomposition only for several seconds. Raising or lowering the temperature increases the time during which austenite exists (incubation period).

At temperatures in the vicinity of point  $A_r$  ( $700^\circ\text{C}$ ) austenite decomposes and pearlite is formed; at lower temperatures ( $600^\circ\text{C}$ ) sorbite is formed and at  $500^\circ\text{--}550^\circ\text{C}$ —troostite. If the temperature is lowered from  $550^\circ$  to  $220^\circ\text{C}$  acicular troostite is formed (it is known as bainite in English and American metallurgical literature). The change in the hardness of the formed structures is shown in Rockwell units ( $R_c$ ) at the right-hand side of the diagram in Fig. 46.

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## THE HEAT TREATMENT OF METALS AND ALLOYS

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### *Chapter 8*

#### HEAT TREATMENT OF STEEL

Steel undergoes heat treatment to improve its structure and to obtain higher or, in any case, specified mechanical properties.

*Heat treatment* is the process of changing the structure of a metal by heating it to a predetermined temperature, holding it at this temperature for a prescribed period of time and cooling it at a prescribed rate.

The types of heat treatment applied in practice are: (1) annealing, (2) normalisation, (3) hardening and (4) tempering.

#### **8-1. Annealing and Normalising of Steel**

**Annealing of steel.** *Annealing* is one of the most important heat treating operations applied to steel. It relieves internal stresses in the metal, refines the grain and reduces the hardness. After annealing, steel becomes more plastic and ductile. Hypoeutectoid steels are heated for annealing 30 to 50 deg C above the critical point  $A_c$ , and hypereutectoid steels the same amount above point  $A_{c1}$  (line *GSK* in the iron-carbon equilibrium diagram). Heating metals to be annealed to higher temperatures is accompanied by the development of a coarse-grained structure and is called *overheating*.

The original structure of overheated steel can be restored by a second annealing operation at the proper temperatures. If the heating temperature in annealing is very high and approaches the solidus (beginning of melting for the alloys) the steel may be *burned*. Here the metal is oxidised on the grain boundaries. Burning is a defect that cannot be remedied.

After the required heating temperature is achieved in annealing, the metal is held for a certain time in the furnace to enable the required changes to take place throughout the mass of the workpiece. This holding time in the furnace at the annealing temperature usually ranges from 30 to 60 minutes per ton of the charge. Excessive holding may lead to overheating of the steel which will acquire a coarse-grained structure.

The metal is commonly cooled, after heating and holding, with the furnace; a cooling rate from 150 to 200 deg C per hour is regular practice (depending upon the amount of carbon and certain other elements in the steel).

If the steel has been subjected to cold working to increase the tensile strength and yield point but has, at the same time, become work-hardened and brittle (lost its ductility), so-called *recrystallisation*, or *process annealing* is applied. Its purpose is to reduce the distortions of the crystal lattice produced by cold working.

Recrystallisation annealing consists in heating the steel to a temperature of about 600-700° C, holding at this temperature for a prolonged period and slow cooling. This treatment transforms the grains of steel, broken up or distorted in the process of deformation, back into their normal state (by recrystallisation) thus removing work-hardening and relieving internal stresses.

Recrystallisation annealing is frequently applied in the production of cold-rolled steel strip, in deep drawing and in wire drawing operations as an intermediate process with the aim of increasing the plasticity of the steel.

According to Academician A. Bochvar the recrystallisation annealing temperature can be determined from the formula

$$T_{rc} = 0.4 T_m, \quad (60)$$

where  $T_m$  = absolute melting temperature of the given metal.

Tempering at temperatures of 300° to 400° C is often substituted for recrystallisation annealing of steel to relieve internal stresses and to reduce volume changes; this will be discussed in more detail further on.

Ordinary annealing is incapable, in many cases, of reducing the hardness of many grades of alloy steel to a sufficient degree. Isothermal annealing is resorted to in such cases.

*Isothermal annealing* comprises two stages. First, the steel is heated above the upper (or only the lower) critical point and held for a certain time at this temperature. At this stage austenite is formed in the metal. Then the steel is rapidly cooled to a temperature 50 to 100 deg C below point  $A_{c1}$ , i. e., to 600-700° C, and is held at this new temperature until the austenite is completely decomposed to form pearlite. Finally, the steel is cooled in still air.

In this procedure austenite decomposition takes place at a constant temperature and hence the name—*isothermal annealing*. A more homogeneous structure of the metal is obtained and the results of annealing are more stable. The cycle of heat treatment is substantially reduced in isothermal annealing.

A granular pearlite structure can be obtained in hypereutectoid steels (usually in medium- and high-carbon tool steels) by annealing to granular cementite, otherwise called *spheroidising*.

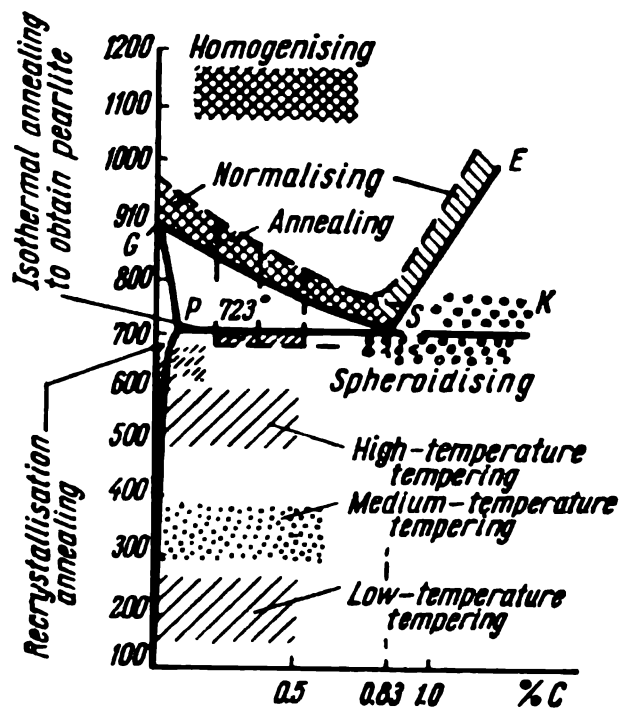


Fig. 47. Temperature ranges for various steel annealing and tempering procedures

*Spheroidise annealing* comprises heating steel to a temperature slightly above point  $A_{c1}$  (to  $730^{\circ}$ - $770^{\circ}$  C) and prolonged holding at this temperature followed by slow cooling (25 to 30 deg per hour) to  $600^{\circ}$  C with the furnace.

Considerable inhomogeneity is always observed in steel castings or ingots. It is associated with the development of dendritic or zonal segregations in the metal. This internal inhomogeneity of the steel can be eliminated or at least reduced to some extent by *diffusion annealing*, or *homogenising*. The procedure involves heating the steel to a high temperature ( $1000^{\circ}$ - $1100^{\circ}$  C), prolonged

holding at this temperature (10 to 15 hrs), subsequent cooling with the furnace to  $800^{\circ}$ - $850^{\circ}$  C and further cooling in still air. Though the composition of the metal becomes more uniform after this heat treatment, a coarse-grained structure is obtained. Therefore, steel castings and ingots are subjected to a second full annealing treatment after homogenising.

**Normalising of steel.** A special highly efficient annealing process, called *normalising*, is sometimes used for the elimination of internal stresses in steel and for its recrystallisation.

*Normalising* is a heat treatment in which the steel is heated 30 to 50 deg C above point  $A_{c1}$ , held at this temperature and then cooled in air. It has essentially the same purposes as conventional annealing and, therefore, the structure of the steel after normalising is similar to that of annealed steel. Hypoeutectoid steels consist of ferrite and pearlite after normalising and hypereutectoid steels—of pearlite and cementite. The normalised structure of low-alloy steel consists of sorbite and ferrite. Normalised steel castings have a higher yield

point, tensile strength and impact strength than if they were annealed. Since normalising requires less time than conventional annealing it is extensively replacing the latter in treating low- and medium-carbon steels. High-carbon steels undergo normalising to eliminate the cementite (carbide) network.

After working (rolling, forging or stamping) certain grades of alloy steel are subjected to normalising to improve their structure, i. e., to bring the grains to their normal condition and to relieve harmful internal stresses in the metal.

The diagram shown in Fig. 47 indicates temperature conditions for various kinds of steel annealing and tempering procedures.

## 8-2. Hardening and Tempering of Steel

The heat-treating process applied to impart high hardness to steel, to improve its mechanical strength and to retain its ductility comprises two operations—hardening and tempering.

**Hardening of steel.** Steel is hardened by heating to a temperature above the upper or lower critical point, holding at this temperature and quenching (rapid cooling). After quenching, a martensite structure is obtained or other transitional forms of austenite, for example, martensite + troostite.

Directly after quenching, the steel acquires a high hardness but, at the same time, high internal stresses develop in the steel and it becomes brittle. The next heat-treating operation — tempering — relieves the internal stresses to some extent and reduces the brittleness of the hardened steel. After hardening and tempering structural steels acquire higher strength, hardness and ductility. The purpose of hardening tool steel is to increase the hardness and wear resistance while retaining sufficient ductility.

The *hardening temperature* depends upon the carbon content of the steel (Fig. 48).

Hypoeutectoid steels, employed chiefly as structural materials in engineering and construction, are heated for hardening 30 to 50

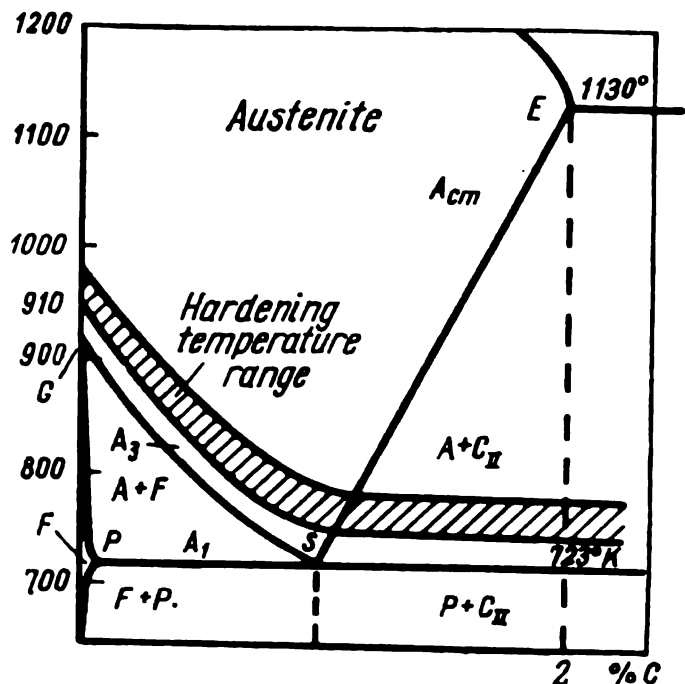


Fig. 48. Temperature range for hardening carbon steels



deg C above point  $A_{c_1}$ . As a result, pearlite and excess ferrite form the homogeneous solid solution, austenite, which is transformed into martensite upon quenching. This procedure is called *full hardening* and it imparts good mechanical properties to the steel if it is followed by a suitable tempering operation.

If hypoeutectoid steel is heated for quenching to a temperature within the interval between the critical points  $A_{c_1}$  and  $A_{c_2}$ , ferrite is partially retained in the steel with the austenite. This ferrite remains unchanged in subsequent quenching and lowers the hardness of the steel. Such a treatment of hypoeutectoid steel is called *incomplete hardening* and finds no application in actual practice since it does not allow the maximum possible strength of the metal to be obtained after tempering.

In hardening hypereutectoid steels they are heated to a temperature 30 to 50 deg C above point  $A_{c_1}$ . This leads only to a partial change in the structure. All of the pearlite and part of the cementite are transformed into the homogeneous solution (austenite) but the rest of the cementite remains in the steel in the form of a carbide network or grains and is retained after quenching. Since hypereutectoid steels are mainly used as the material for tools which require not only high hardness but high wear resistance as well, the presence of cementite in the martensite after quenching is both desirable and expedient. It is only necessary to make sure that the excess cementite, not dissolved in the heating process, is in the steel in the form of separate small grains in the martensite matrix and not as a coarse network.

*Steel hardening methods* include the two principal operations: heating and quenching. If these operations are properly conducted the required structure is obtained and, consequently, the specified physico-mechanical properties.

The heating method, holding time at the proper temperature and the arrangement of the workpieces in the furnace during heating are of great importance.

Steel parts may be heated for hardening in flame furnaces fired with oil, gas or coal, in which the metal is in direct contact with the flame; in muffle furnaces where the metal parts are held in a compartment heated from the outside by flame or electric current and in bath-type furnaces where the workpiece is immersed in molten salts or lead.

Workpieces should be heated to the prescribed temperatures gradually and uniformly to avoid the development of internal stresses in the metal. At the same time, excessively slow heating reduces the output of the furnaces and may lead to decarburisation and oxidation of the surfaces of the steel.

Heating procedure is determined by the shape and size of the workpiece cross section, chemical composition of the steel and the method

of supplying the heat. The more carbon and alloying elements in the steel and the more complex the shape of the work, the slower it should be heated.

The metal is more rapidly and uniformly heated in molten lead or salt baths. Less decarburisation and oxidation is observed in such furnaces. Heat transfer to the work is slower in muffle furnaces than in the flame or bath types.

Controlled or protective atmospheres are introduced into some furnaces to avoid decarburisation and oxidation of the metal. The workpiece surfaces are clean and bright after heating in these furnaces. Such atmospheres consist of dried gaseous products of the incomplete combustion of fuels, purified nitrogen or other gases.

The heating rate in steel hardening and the holding time in the furnace depend upon the chemical composition of the metal, and the shape and cross section of the work. These data are given in special tables which are drawn up on the basis of experience obtained in a number of industrial plants.

The structure of steel is affected by the cooling rate; this determines whether a martensite, troostite or sorbite structure will be obtained after quenching. Consequently, special quenching media are used that enable the rate of the cooling process to be controlled after the steel has been heated to the required temperature.

Of prime importance is the ability of a quenching medium to ensure a high cooling rate in the interval from  $650^{\circ}$  to  $550^{\circ}$  C, i. e., in the range where austenite most rapidly decomposes into a mixture of ferrite and cementite. On the other hand it is necessary to slow down the cooling rate somewhat in the range from  $300^{\circ}$  to  $200^{\circ}$  C within which martensite is formed (martensitic transformation) since the latter process is accompanied by an increase in the volume of the metal, development of high internal stresses and the appearance of hardening cracks.

The most intensive cooling action is displayed by cold water which cools steel at a high rate, not only in the range from  $650^{\circ}$  to  $550^{\circ}$ , but in the  $350^{\circ}$ - $250^{\circ}$  C interval as well.

Weak aqueous solutions of salts, acids and alkalis at room temperature have approximately the same effect as water in quenching steel. The quenching power of water and various aqueous solutions is perceptibly reduced when they are heated. The effect of this factor on the cooling rate is especially great in the upper temperature interval ( $650^{\circ}$ - $550^{\circ}$ ).

Quenching media with less intensive action are mineral and vegetable oils which have a low sensitivity to changes in temperature.

Molten salts of potassium and sodium, as well as molten lead, have the lowest heat-extracting power of the liquid quenching media employed in steel hardening. Their application leads to the formation

of troostite and even sorbite in the metal which substantially reduce its hardness before tempering. The data on the quenching power of various fluids, listed in Table 2, were obtained in the investigations of S. Steinberg.

Table 2  
Cooling Rate of Steel in Various Quenching Media,  
Degrees per Second

Quenching medium	Temperature interval	
	650°-550° C	300°-200° C
Water at a temperature of		
18° C . . . . .	600	270
26° . . . . .	500	270
50° . . . . .	100	270
74° . . . . .	30	200
10% aqueous solution at 18° C		
of caustic soda . . . . .	1200	300
common salt . . . . .	1100	300
soda . . . . .	800	270
Soluble oil in water . . . . .	70	200
Soapy water . . . . .	30	200
Mineral machine oil . . . . .	150	30
Transformer oil . . . . .	120	25

Whenever steel is quenched in water, various aqueous salt solutions or oil, an insulating vapour blanket, or film, envelopes the work thus impeding heat extraction from the metal. Agitation of the parts in the quenching bath breaks up the vapour blanket, thereby allowing intimate contact between the metal and the quenching medium. This increases the cooling rate and leads to more intensive hardening of the steel.

Steel hardening methods are classified as follows according to the quenching procedures applied:

1. *Conventional hardening* (quenching in a single medium) consists in simply immersing the parts in a liquid where they are cooled to room temperature; the cooling effect is intensified by violent agitation of the workpieces in the quenching medium.
2. *Quenching in two different media*. Steel is rapidly quenched in the first liquid, usually water, to a temperature from 300° to 400° C and then is quickly transferred to a quench bath of less severity (oil, as a rule) where it is held until it is completely cooled. This procedure relieves internal stresses in the metal. It is most frequently employed for the heat treatment of tools made of carbon steels.
3. *Stepped quenching* is also done in two stages. After heating to the required temperature above line GSK, the steel is first immersed in a quenching medium having a temperature from 150 to 300 deg

C above point  $M_s$  of the martensitic transformation for the given steel composition (see Fig. 46). The part is held until it reaches the temperature of the medium, usually a molten salt bath, throughout its whole cross section. Then the part is transferred to an oil bath where it cools to room temperature.

Comparatively low internal stresses are developed in stepped quenching but parts with large cross sections cannot be hardened by this method because they do not achieve the critical cooling rate in the first quench. A structure of martensite with retained austenite is obtained after 'stepped quenching.

4. *Interrupted quenching, or martempering*, is a modification of stepped quenching. Here, after quenching in a liquid medium and holding for a definite time, the work is removed from the bath still hot and allowed to cool further in still air to room temperature. The martensitic transformation takes place under conditions of a lower cooling rate and therefore the internal stresses are reduced to a still greater extent.

5. *Isothermal quenching, or austempering*, is performed so that the austenite decomposes at a constant temperature slightly above that at which the martensitic transformation begins (point  $M_s$  in Fig. 46). In austempering, the steel part is heated to the required hardening temperature and then quenched in a medium having a temperature from 300° to 250° C, where it is held for as long a time as is needed for the isothermal transformation of the austenite. Since the quenching bath temperature is higher than the  $M_s$  temperature, the austenite is transformed into acicular troostite which has a sufficiently high hardness in conjunction with ample ductility.

The main advantages of austempering are that the residual stresses in the metal are substantially reduced; there is less danger of hardening cracks being formed and less warpage.

Salt and alkali quenching baths of various compositions ( $\text{NaNO}_3$ ,  $\text{KNO}_3$ ,  $\text{KOH}$ ,  $\text{NaOH}$ , etc.) are common for austempering. The melting temperature for salt mixtures ranges from 130° to 220° C.

6. *Surface hardening* is a selective heat treatment in which a high hardness is imparted to the surface layers of the work whilst the tough core is retained. Only the surface layers of the steel part are heated to the hardening temperature and it is then quenched in water or some other medium. The most widely used heating methods for surface hardening are by induction with a high-frequency current and by means of gas flame burners.

The heating principle in induction hardening is based on the current induced in the surface layers of a part placed inside the coils of an inductor through which a high-frequency current flows. The induced current passes chiefly along the surfaces of the part, due to the so-called skin effect, and heats the metal to a high temperature.

The depth to which the induced current penetrates the metal and, consequently, the depth to which the part is heated before quenching depends upon the frequency of the primary current in the inductor, and the values of the magnetic permeability and electrical resistivity of the steel.

The depth to which the metal is heated and the corresponding depth of the hardened layer can be controlled by varying the frequency of the primary current and the time the work is held in the inductor. At a given frequency the depth of the hardened layer increases with the holding time in the inductor. The higher the current frequency, the less the current penetration into the part will be. The induced current is distributed nonuniformly over the cross section of the work; the current density is considerably higher at the surface than in the core.

The depth of current penetration can be determined by the following empirical formula:

$$\delta = \frac{500}{\sqrt{f}} \text{ cm,} \quad (61)$$

where  $f$  = current frequency, in cps.

Either machine or vacuum-tube type generators supply the high-frequency current for induction hardening of steel. The principle of a high-frequency unit with a motor (machine-type) generator for induction heating of steel is illustrated in Fig. 49. The work to be surface-hardened is heated within the inductor.

In flame hardening, the surface of steel is usually heated by an oxy-acetylene flame whose temperature may reach 3000° C. The gas burner with the oxy-acetylene flame travels along the work at a definite speed (about 150 mm per min) and heats its surface to the required temperature and depth. The burner is followed by a traveling stream of cold water which quenches the heated layer of metal and thereby hardens it.

**Subzero treatment.** A definite amount of austenite that did not have sufficient time to transform into martensite remains after quenching certain grades of steel. This retained austenite reduces the hardness, wear resistance and certain other properties of steel. These drawbacks can be eliminated by subjecting the steel to a subzero treatment.

Subzero treatment consists in cooling the steel part, after hardening, to a subzero temperature. As a result, a considerable part of the retained austenite is transformed into martensite.

Subzero treatment is most effective for steels containing more than 0.6-0.7% C which retain a large amount of austenite after quenching. The end of the martensitic transformation (point  $M_f$  in Fig. 46) for these steels lies in the subzero temperature region. Certain prop-

erties of these steels can be changed if they undergo subzero treatment directly after quenching.

At properly assigned temperatures subzero treatment considerably increases the hardness and improves the cutting ability of carbon and high-speed steel tools and is effective in stabilising the dimensions of gauges and other measuring tools.

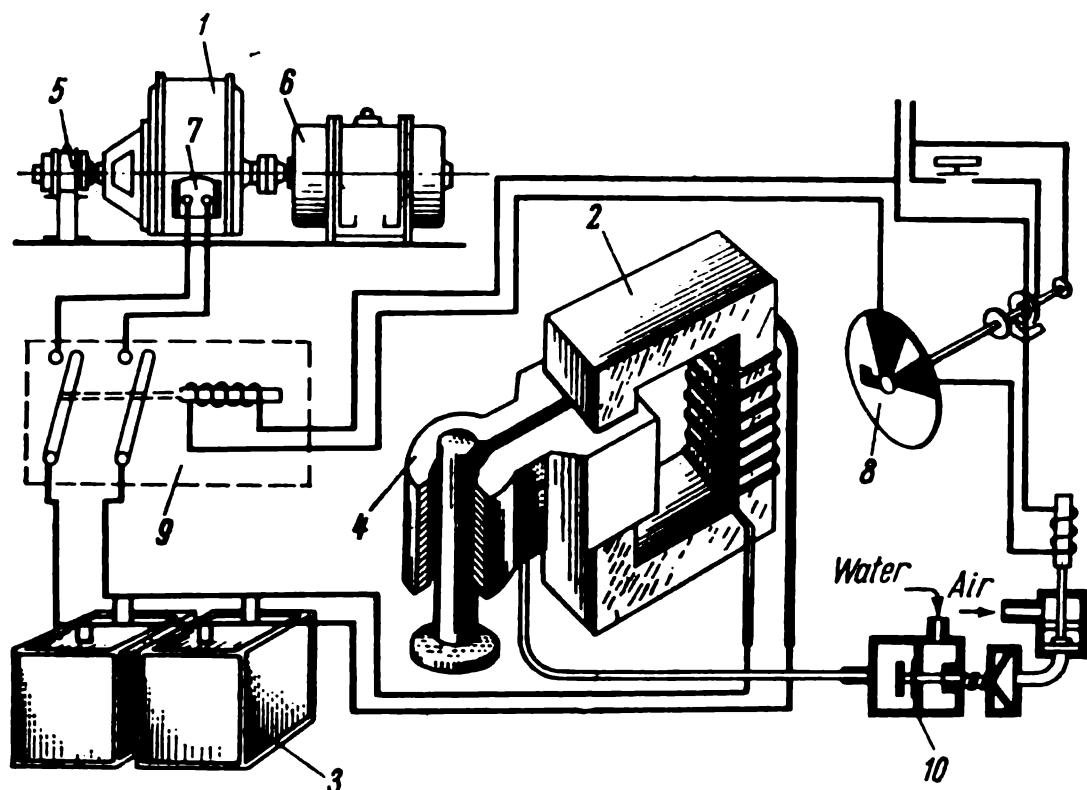


Fig. 49. Diagram of a high-frequency unit with a motor generator for the induction heating of steel:

1 — generator; 2 — transformer; 3 — capacitor bank; 4 — inductor; 5 — exciter; 6 — motor; 7 — starter; 8 — time-lag relay; 9 — contactor; 10 — solenoid-controlled pneumatic-hydraulic valve

The subzero treatment of steel is performed in units which produce subzero temperatures in a range from  $-75^{\circ}$  to  $-195^{\circ}$  C.

**Tempering of steel.** All hardened parts undergo subsequent tempering. *Tempering* is a heat treatment in which hardened steel is heated to a temperature below the critical point  $A_{c1}$ , held for the proper time at this temperature and then slowly cooled.

The purposes of tempering are to reduce the internal stresses to some extent, stabilise the structure of the metal, reduce the hardness and increase the ductility. The structural transformations in tempering involve decomposition of the martensite and retained austenite which form more stable structures.

Tempering temperatures are assigned on the basis of martensite decomposition data. It has been found that no detectable structural

changes take place when hardened steel is heated up to 100° C. In the range from 100° to 350° C martensite decomposes with the precipitation of cementite (iron carbide) particles. This decomposition comprises two stages. In the first stage, at temperatures from 100° to 200° C, fine carbide inclusions separate from the martensite. These inclusions are most frequently of plate form with a thickness of several atom layers. At the same time there is a noticeable decrease in the volume of the steel. This is associated with the formation of a new structure—tempered martensite—characterised by a reduction in the crystal lattice constant. The second stage in tempering occurs in the temperature range from 200° to 350° C. It involves a further slow precipitation of iron carbide particles from the martensite. Decomposition of the retained austenite into tempered martensite takes place in this second stage as well.

As a rule, retained austenite decomposition in the tempering process is completed at 350° C. As a result of the decomposition of martensite and retained austenite, steel tempered in the range from 300° to 350° C acquires a secondary troostite structure which is a finely dispersed mixture of ferrite and cementite.

Upon further heating, the carbide particles are coalesced and acquire a globular form. Up to temperatures of about 500° C, the particles retain their plate form; above 500° they are spheroidised into globules. Tempering at temperatures in the interval from 500° to 600° C leads to the formation of a sorbite structure.

Tempering is classified into three types according to the heating temperatures whose ranges are specified in the lower part of Fig. 47. They are:

1. *Low-temperature tempering* which is done in the temperature interval from 150° to 250° C. The purpose of this procedure is to relieve the internal stresses and to increase the ductility without essentially changing the structure of the steel or reducing its hardness. Low-temperature tempering is applied in the heat treatment of carbon and low-alloy steel cutting tools, as well as measuring tools and parts that have been carburised and surface-hardened.

2. *Medium-temperature tempering* which involves heating the work to 350°-450° C. The structure of steel is altered by this procedure; martensite is transformed into troostite. The results are a reduction in hardness and strength of the metal and an increase in the elongation and ductility.

3. *High-temperature tempering* which is performed in the range from 500° to 650° C. At these temperatures sorbite is formed in the steel and the internal stresses are almost completely eliminated. A sorbite structure provides the most expedient proportion between the strength and ductility of structural steels in whose heat treatment high-temperature tempering is applied.

**Hardenability of steel.** Hardenability is the property of steel to harden to a definite depth. Parts of carbon steel up to 25 mm in diameter can be through hardened. Parts of larger diameter do not possess this ability for through hardening; they comprise two zones: an external hardened zone and an unhardened core.

The depth of hardening is defined as the distance from the surface of a part to a layer comprising 50 per cent martensite and 50 per cent troostite. This distance is determined by making hardness tests along the cross section of a hardened specimen. Hardness depends chiefly upon the amount of martensite; the more the martensite, the higher the hardness of the specimen will be.

Hardenability depends upon the composition of the steel and the physical properties of the quenching medium. In assigning a definite grade of steel, the designer must make his decision with due consideration for hardenability data. He must know whether the whole mass of the part will harden to martensite or only to a certain depth from the surface since the mechanical properties of parts hardened to a definite depth are nonuniform.

The hardenability of alloy steels is higher than that of the carbon grades and therefore they are more frequently used in the more critical designs.

**Hardening defects.** Improperly conducted hardening procedures may be the cause of various defects in the metal, which include insufficient hardness, soft spots, excessive brittleness, deformation, warping and cracks.

*Insufficient hardness* may be due to:

- (1) low hardening temperature for hypoeutectoid steels;
- (2) insufficient holding time at the hardening temperature;
- (3) low cooling rate in quenching;
- (4) overheating of hypereutectoid steels before quenching.

The remedy for this defect is steel normalisation or annealing which is to be followed by a new hardening operation conducted strictly according to the prescribed heating and quenching procedures.

*Soft spots*, i. e., unhardened regions or ones with a lower hardness, are the result of an inhomogeneous initial structure or decarburisation of the surface of the work. Aggregations of ferrite in the initial structure, for example, do not have time to pass into the solid solution during hardening and are retained after quenching. The formation of decarburised areas on the surface of a part naturally affects the hardness of the metal after quenching.

Inhomogeneity of the initial structure can be eliminated by annealing or normalising the steel before hardening. The use of a more severe quenching medium also prevents the formation of soft spots.

*Excessive brittleness* is developed in steel when it is quenched from a higher temperature than is required which leads to austenite grain



growth. This defect is revealed in examining the microstructure or a fracture of the steel or it becomes evident in mechanical tests. Excessive brittleness is eliminated by hardening the steel a second time and following the proper heating procedure.

*Deformation, warping and cracks* are the result of volume changes and internal stresses developed in the metal during quenching.

Deformation of the work is due to the martensitic transformation in steel after quenching which is associated with a certain increase in the volume of the metal. Warping is due to nonuniform heating or cooling of the work. Both defects can be prevented or, at least, reduced by uniform heating and cooling, by the application of quenching presses or, finally, by straightening and grinding the part after hardening.

*Cracks* are caused, not only by the internal stresses that develop in steel during quenching, but also by the structural transformations that occur. Cracking can be prevented by avoiding abrupt changes in the cross section when designing a part and by annealing the part before machining to relieve internal stresses. Martempering and stepped quenching also assist in preventing quenching cracks.

## Chapter 9

### CHEMICAL HEAT TREATMENT OF STEEL (CASEHARDENING)

The saturation of the surface of steel with certain elements from the surrounding medium by their diffusion at high temperatures changes, not only the chemical composition, but also the structure and the properties of the surface layers of the part. This operation is called *chemical heat treatment* or, more simply, *casehardening* and its purpose is to increase the surface hardness of steel, its wear resistance, etc.

Casehardening processes are classified as: (1) carburising (saturation with carbon); (2) nitriding (saturation with nitrogen); (3) cyaniding and carbonitriding (enrichment with both carbon and nitrogen) and (4) diffusion coating (impregnation with other metals).

#### 9-1. Carburising of Steel

*Carburising* is the process of saturating the surface layer of low-carbon steels at 850°-950° C with carbon from a carbonaceous source capable of giving up its carbon to the metal.

Such carbonaceous materials are called *carburisers*.

The depth of carbon penetration in carburising steel ranges from 0.5 to 2 mm, and the carbon content in the surface layer is increased to 0.75-1.2 per cent.

There are three methods of carburising, namely: pack carburising (using a solid carburiser); liquid carburising and gas carburising.

*Solid carburisers* are powdery mixtures consisting of charcoal (about 70 per cent by weight), barium or sodium carbonate (about 20 to 25 per cent by weight) and calcium carbonate (up to 2.5-3.5 per cent by weight). The parts to be carburised are packed in this compound in a steel box.

Usual practice is to cover the bottom of the box with a 25 to 30 mm layer of the solid carburiser on which the bottom row of parts is laid. This row is covered by a new layer of carburiser 15 to 20 mm thick which is then rammed before laying the next row of parts, and so on until the box is filled. A space of 15 to 25 mm is left between the parts and walls of the box. The top row of parts is covered with 30 to 40 mm of carburiser and then the box is closed with a steel cover. The joint between the box walls and cover is luted with fire clay to prevent the penetration of air and furnace gases. The packed boxes are charged into a furnace and heated to approximately 900°-950° C. Heating time to the carburising temperature depends on the weight of the charge, while holding time depends upon the depth of the carburised layer required.

The depth of carbon penetration is about 0.1 mm per hour of holding time in the furnace at the carburising temperature.

The pack carburising of steel is a complex process and actually proceeds with the participation of a gaseous phase formed in the box by the action of the high temperatures.

There is always air present in the carburising box; it has been absorbed by the charcoal and fills the space between the grains of carburiser. When the box is heated to 900°-950° C, the oxygen in the air and the carbon in the charcoal react to produce carbon monoxide according to the equation  $2C + O_2 = 2CO$ . In the presence of iron, a certain portion of the carbon monoxide dissociates forming atomic carbon ( $2CO = CO_2 + C_{at}$ ).

Atomic carbon evolved in this last reaction is in the nascent state and therefore highly active. It easily diffuses into the gamma iron, saturating it with carbon even up to the formation of cementite in the surface layers according to the reaction



The carbonates added to the charcoal are activators and accelerate the carburising process since they regulate the pressure and composition of the gaseous phase inside the box. Barium carbonate, for example, decomposes at high temperatures according to the equation

$BaCO_3=BaO+CO_2$ , thereby supplying carbon dioxide. The latter, in turn, reacts with the charcoal ( $CO_2+C=2CO$ ) and thus replenishes the supply of carbon monoxide inside the box.

Carburised steel parts are subjected to subsequent heat treatment, comprising hardening and tempering.

*Gas carburising* is accomplished by passing gases capable of forming atomic carbon at high temperatures over parts made of mild steel. Such carbon-producing gases include: natural gas, city gas and artificially produced mixtures of methane, ethane, propane and other gases obtained in the cracking of kerosene and other liquid petroleum products.

Gas carburising has a number of advantages over pack carburising. In the first place, less time is required for the operation and it is possible to control the process of saturating the steel with carbon more accurately. Gas carburising excludes the necessity of preparing the carburiser and packing and unpacking the boxes. Consequently, it becomes feasible to mechanise the process and to combine carburising with subsequent hardening of the work.

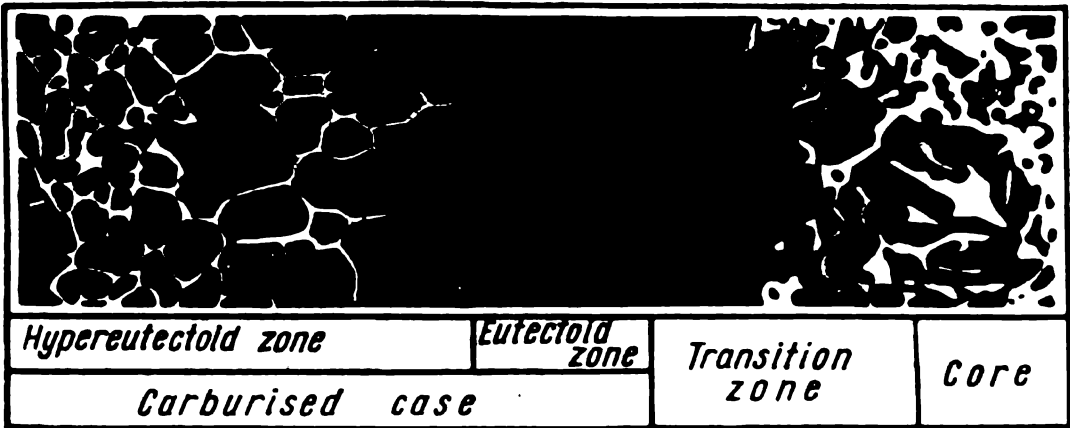


Fig. 50. Zones in the microstructure of carburised steel

In practice, gas carburising is carried out in the following manner. The parts to be carburised are charged into air-tight chambers of batch-type or continuous furnaces and are heated to 900°-950° C. Then the carburising gas is passed through the chamber for a period of 6 to 10 hours.

A high-speed gas carburising process developed in the U.S.S.R. consists in induction heating of the parts by high-frequency current to a temperature of 1080°-1100° C. These conditions enable a carburised layer 0.8 to 1 mm deep to be produced in 40 to 50 minutes. The work is quenched directly after cooling it to 870° C.

*Liquid carburising* is accomplished by several methods. The most suitable liquid carburising medium is a bath of molten salts con-

sisting of a mixture of sodium carbonate (80 per cent) and sodium chloride (10 to 15 per cent) to which silicon carbide (6 to 10 per cent) has been added.

The reaction in the bath



leads to the saturation of the metal with carbon. The composition of the bath is renewed from time to time. Parts immersed in the bath at a temperature from 870° to 900° C acquire a carburised layer 0.2 mm deep in 40 to 50 minutes.

The principal advantages of liquid carburising are the uniform heating of the work and the possibility of quenching direct from the bath.

Fig. 50 shows the structure of the carburised layer, or case, in steel that was slowly cooled from the carburising temperature. The carbon content in the carburised layer decreases gradually in the direction from the surface to the core.

## 9-2. Nitriding of Steel

*Nitriding* is the process of saturating the surface of steel with nitrogen. It is accomplished in an atmosphere of ammonia gas ( $\text{NH}_3$ ) at temperatures ranging from 480° to 650° C. Ammonia is almost completely dissociated in this temperature interval according to the equation  $2\text{NH}_3 = 2\text{N} + 3\text{H}_2$ . Atomic nitrogen formed in ammonia dissociation diffuses into the alpha iron and saturates the metal. After nitriding, the work is cooled down to 200° C in a stream of ammonia.

Nitrided steel acquires a very high surface hardness which is retained even after subsequent heating up to 600°-650° C. Nitriding increases the wear resistance of steel, its fatigue limit and its corrosion resistance in air, water and water vapour.

Nitriding is usually applied to medium-carbon and alloy steels containing aluminium, chromium, molybdenum and other elements capable of forming nitrides ( $\text{Fe}_3\text{N}$ ,  $\text{Fe}_4\text{N}$ ,  $\text{Cr}_3\text{N}$ ,  $\text{Mo}_3\text{N}$ , etc.). Since nitriding is done at low temperatures it requires more time than gas carburising, for example.

Prior to nitriding, parts should be hardened, tempered and undergo the complete sequence of machining operations, including grinding. Only finish grinding or lapping is done after nitriding. The nitrided case is usually from 0.2 to 0.4 mm deep.

Nitriding can be conducted at temperatures far below those required for hardening and carburising. No quenching is necessary as the high hardness is obtained directly after the operation. This feature enables hardening defects to be avoided.

### 9-3. Cyaniding and Carbonitriding of Steel

*Cyaniding and carbonitriding* are casehardening processes in which both carbon and nitrogen are added to the surface layer of steel. The case obtained is shallow; its depth ranges from 0.1 to 0.2 mm. These processes increase the surface hardness, wear resistance and fatigue limit. They are especially effective for medium and small parts, such as gears, pistons, pins, small shafts, etc.

Both of these casehardening procedures are based on the decomposition of cyanide compounds that easily release the cyan group (CN). Since the cyan group contains carbon and nitrogen atoms, iron is simultaneously saturated with both.

Cyaniding involves heating the steel in a liquid or solid medium; if the process is performed in a gas atmosphere it is called carbonitriding.

*Pack cyaniding* is accomplished in a powdery mixture consisting of 60 to 80 per cent charcoal and 40 to 20 per cent potassium ferrocyanide  $K_4Fe(CN)_6$ . The cyaniding temperature is from  $540^\circ$  to  $560^\circ$  C and the holding time is from 1.5 to 3 hours. This method can be employed under any shop conditions, using the simplest equipment. At the present time it is chiefly used to improve the cutting properties of tools.

*Liquid cyaniding* is more widely applied. It is done in a bath of molten neutral salts ( $Na_2CO_3$ , NaCl, etc.) in which various cyanide compounds NaCN,  $Ca(CN)_2$ , etc., are dissolved.

Liquid cyaniding processes may be classified into three groups: low-, medium- and high-temperature cyaniding.

The bath for low-temperature cyaniding contains 25 to 40% NaCN, 20 to 45%  $Na_2CO_3$  and 10 to 20% NaCl. Work is cyanided at a temperature of  $550^\circ$ - $600^\circ$  C and the holding time ranges from 5 to 30 minutes, depending upon the required depth of the cyanided case. The case obtained in this procedure has a depth from 0.02 to 0.04 mm and is characterised by its high hardness, good wear resistance and low friction in machining. Low-temperature cyaniding is employed mainly for treating high-speed steel tools as it increases tool life considerably.

A bath for medium-temperature cyaniding contains up to 20-35% NaCN and is heated to  $800^\circ$ - $850^\circ$  C. The holding time in the bath is from 5 minutes to 1.5 hours depending upon the case depth required. The resulting case (0.075 to 0.25 mm deep) contains 0.6 to 0.8% C and 0.4 to 0.5% N. The case has a Rockwell hardness (Scale C) ranging from 56 to 60.

Medium-temperature cyaniding is followed by quenching and tempering.

Many machine parts made of medium-carbon steels, such as bolts, nuts, small gears, etc., undergo medium-temperature cyaniding.

High-temperature cyaniding is carried out at 900°-950° C in baths containing 6 to 10% NaCN, 80 to 84% BaCl<sub>2</sub>, and up to 10% NaCl. Case depth depends upon holding time which is within 1.6 hours.

High-temperature cyaniding produces a case from 0.5 to 1.5 mm deep containing from 1 to 1.2% C and 0.2 to 0.3% N. This process is followed by quenching and low-temperature tempering (160°-180° C).

Extreme care and strict observance of the safety rules are of prime importance in the operation of cyaniding baths due to the high toxicity of the cyanide salts.

*Carbonitriding* consists in treating steel parts with a mixture of carburising and nitriding gases. A mixture of city (or natural) gas (70 to 80 per cent) and ammonia (20 to 30 per cent), for example, may serve for this purpose.

In carbonitriding the surface of the steel is also saturated with carbon and nitrogen simultaneously but the process is slower than in liquid cyaniding.

Carbonitriding can be conducted at various temperatures from 850° to 930° C depending upon the type of heating furnace used, the grade of steel being treated and the specified case depth.

#### 9-4. Diffusion Coatings

*Diffusion coating*, or metallic cementation, is the process of impregnating the surface of steel with aluminium, chromium, silicon, boron, beryllium and other elements.

Diffusion coating is accomplished by heating and holding steel parts in direct contact with one of the above elements which may be in the solid, liquid or gaseous state.

This process imparts a number of valuable properties to steel, among which are high heat, corrosion and wear resistance. In many cases, steel subjected to diffusion coating may be used as a substitute for a high-alloy steel.

*Calorising*, the surface impregnation of steel parts with aluminium, is done to prevent scaling at elevated temperatures (up to 850°-900° C). Upon heating calorised steel a dense surface film of aluminium oxide (Al<sub>2</sub>O<sub>3</sub>) is formed which provides the protective action against oxidation.

In powder calorising the parts are first packed in a box where they are embedded in aluminium (or ferroaluminium) powder with an admixture of 0.5 to 2 per cent ammonium chloride (NH<sub>4</sub>Cl) and then the box is heated to 950°-1000° C. The holding time depends upon the case depth required.

Dip calorising consists in the immersion of steel parts into a bath of molten aluminium heated to a temperature of  $750^{\circ}$ - $800^{\circ}$  C. The process of impregnating the steel with aluminium takes from 45 to 90 minutes and a calorised layer 0.2 to 0.35 mm deep is obtained.

Gas calorising is done in closed retorts with vaporous aluminium chloride ( $\text{AlCl}_3$ ) which undergoes partial dissociation at high temperatures and impregnates the steel with aluminium.

Steel calorised by any of the three foregoing methods is subjected to subsequent diffusion annealing at  $900^{\circ}$ - $1000^{\circ}$  C to reduce the brittleness of the calorised layer.

*Chromising* is a process in which the surface of steel parts is impregnated (alloyed) with chromium from either a solid, liquid or gaseous medium.

Pack chromising is the simplest of these techniques. It is accomplished by packing the work in a box containing a powdery mixture of finely ground ferrochromium (40 to 45 per cent), chamotte (45 to 50 per cent) and ammonium chloride (3 to 5 per cent). The processing temperature is from  $1100^{\circ}$  to  $1150^{\circ}$  C, and from 10 to 15 hours of holding time is required.

For liquid chromising the steel parts are heated to  $900^{\circ}$ - $1100^{\circ}$  C in a bath consisting of the molten chlorides of barium, magnesium and calcium ( $\text{BaCl}_2$ ,  $\text{MgCl}_2$  and  $\text{CaCl}_2$ ) to which either 15 to 30 per cent chromium chloride ( $\text{CrCl}_3$ ) or 15 to 25 per cent ferrochromium is added.

A more effective method is gas chromising of steel. Here the parts are heated to  $950^{\circ}$ - $1050^{\circ}$  C in an atmosphere of vaporous chromium chloride ( $\text{CrCl}_3$ ) obtained by passing a mixture of hydrogen and hydrochloric acid vapour through chromium or ferrochromium powder at high temperatures.

Chromised steels become scale resistant (especially at temperatures of up to  $800^{\circ}$ - $900^{\circ}$  C), and possess good corrosion resistance to aqueous solutions of certain acids and to sea water.

*Siliconising*, a metallic cementation process involving silicon, increases the wear resistance of steel and improves its corrosion and acid resistance as well.

In pack siliconising, the steel parts are heated to  $1100^{\circ}$ - $1200^{\circ}$  C in a powdery mixture of ferrosilicon (up to 60 per cent), alumina or kaolin (up to 38-39 per cent) and ammonium chloride (1 or 2 per cent). The depth of the siliconised layer is a function of the holding time and ranges from 0.2 to 0.8 mm.

Gas siliconising is done in retort furnaces in which the steel parts are heated in an atmosphere of silicon tetrachloride ( $\text{SiCl}_4$ ) obtained by passing chlorine over ferrosilicon or silicon carbide (carborundum).

The parts are held in the furnace at  $950^{\circ}$ - $1050^{\circ}$  C for from 2 to 4 hours. This produces a case from 0.5 to 1.24 mm deep.

Silicon concentration in the surface layer is as high as 14 or 15 per cent.

Metallic cementation with other elements (beryllium or boron) has not yet found wide application though it is promising.

## Chapter 10

### CLASSIFICATION AND APPLICATIONS OF STEELS

Commercial grades of steel are classified, not only according to their method of manufacture—basic and acid open-hearth, Bessemer, Thomas, and basic and acid electric-furnace steels—but also according to their chemical composition, uses for which their properties fit them and their structure.

As to their composition steels are classified as carbon and alloy steels.

#### 10-1. Carbon Steels

Carbon steels contain up to 2% C and definite amounts of inevitable impurities (or minor constituents, as they are called) which include silicon, manganese, sulphur and phosphorus.

Minor constituents have a negligible effect on the properties of steel when their content does not exceed: 0.3-0.4% Si, 0.5-0.8% Mn, 0.08% P and 0.04% S.

Phosphorus and sulphur are harmful impurities of steel.

*Phosphorus* forms a chemical compound, iron phosphide  $\text{Fe}_3\text{P}$ , and solid solutions with iron. The presence of phosphorus in steel imparts cold-shortness, i. e., high brittleness at normal and especially at low temperatures. It increases the tensile strength but, at the same time, reduces the impact strength and the ductility; at a phosphorus content of slightly over 0.2 per cent, the impact strength has a zero value. The embrittlement effect of phosphorus increases with an increase in the carbon content of the steel.

*Sulphur* forms the chemical compound  $\text{FeS}$  (36.4% S) with iron which is practically insoluble in solid iron. The eutectic ( $\text{Fe} + \text{FeS}$ ) formed in the alloying of iron and sulphur has a melting point of  $985^\circ\text{C}$ . Sulphur is the cause of red-shortness in steel. Red-shortness, i. e., increased brittleness at elevated temperatures, in the presence of sulphur is due to the fusible eutectic (m. p.— $985^\circ\text{C}$ ) formed at the grain boundaries. This eutectic melts when steel is heated for hot-working (forging, rolling, etc.) and breaks up the bonds between the steel grains. The presence of a substantial amount of sulphur (sulphur



inclusions) in steel unfavourably affects other properties. Thus, sulphur noticeably reduces the mechanical properties of steel (tensile strength, yield point and fatigue limit) and decreases its corrosion resistance.

The content of phosphorus and sulphur is strictly limited because of their harmful effect on the quality of steel.

According to the purpose for which they are made carbon steels are classified in the U.S.S.R. as: (1) ordinary structural steels, (2) quality structural steels and (3) tool steels.

*Ordinary carbon structural steels* are used in the as-received condition, without subsequent heat treatment. They are further classified into steels of group A, which are supplied with their mechanical properties guaranteed, and group B in which the chemical composition is guaranteed. Ordinary-quality steels are identified by the letters Ст. (the first two letters of the Russian word for steel) followed by the numerals 1, 2, 3, etc., for example, Ст. 1, Ст. 2, Ст. 3, Ст. 4 and Ст. 5. Increasing numbers indicate an increase in carbon content and tensile strength and a decrease in ductility.

Ordinary carbon structural steels are available as rolled stock—sheets, plates, angles, beams, bars, etc.

*Quality carbon structural steels* undergo subsequent heat treatment; they are widely employed in the engineering industries and are supplied in the form of sheets, plates, bars, forgings and other semifinished products.

Soviet steels of this type are identified by the numbers 08, 10, 20, 25, etc., up to 75 which indicate the average carbon content in hundredths of one per cent. Steels with an increased manganese content (0.8 to 1 per cent) are identified as 15r, 20r, 30r, 40r, 50r, 60r, and 70r.

*Carbon tool steels* acquire high hardness ( $R_C=62-65$ ) and wear resistance after hardening and low-temperature tempering (at 160°–250° C); their hardenability is not very high.

A shortcoming of carbon tool steel is the considerable loss in hardness when it is heated above 225° C. Cutting tools of carbon steel can only be used when their cutting edges are not heated to high temperatures, i. e., for low cutting speeds and for machining soft materials.

Carbon tool steels are designated by: Y7 (Y7A), Y8 (Y8A), Y9 (Y9A), Y10 (Y10A), Y11 (Y11A), Y12 (Y12A) and Y13A where Y is the first letter in the Russian word for carbon, the number is the average carbon content in tenths of one per cent and the letter A indicates that the steel is of high quality.

Steels Y7 and Y7A are suitable for tools requiring high toughness and moderate hardness (cold chisels, forging dies, rivet sets, screw drivers, shears, sledge and machinist's hammers and carpenter's tools).

Tools subject to impact in operation and requiring increased hardness can be more expediently made of steel Y8 or Y8A. Such tools include simple dies, drifts, centre punches, saws, cutting tools for copper, etc. Hardness, in conjunction with a certain degree of toughness, is characteristic of tools made of steel Y9 and Y9A (hollow punches, centre punches, woodworking tools, etc.). Steels Y10 and Y11 or Y10A and Y11A are used for tools not subject to sudden heavy blows and in which a certain toughness is required on the sharp edges.

Tools for impactless operation, whose chief requirement is a very high hardness can be manufactured of steel Y12 and Y13 or Y12A and Y13A.

## 10-2. Alloy Steels

Alloy steels find more and more extensive application each year.

A steel is said to be *alloyed* when its composition incorporates specially introduced (alloying) elements, absent in straight carbon steel, or when the silicon and/or manganese content exceeds the usual percentage.

The most frequently employed elements for alloying steel are Cr, Mn, Si, Ni, W, Mo, Ti, Al, Nb and Zr.

Alloy steels can be classified according to their chemical composition, structural class and purpose.

**Classification according to chemical composition.** In this aspect alloy steels are divided into three-component steels, containing one alloying element in addition to Fe and C; four-component steels, containing two alloying elements, etc.

A system of Russian letters and numerals is used in the U.S.S.R. for the designation of alloy steels. The letters indicate the presence of definite alloying elements, namely: Г—manganese, С—silicon, Х—chromium, Н—nickel, М—molybdenum, В—tungsten, Φ—vanadium, Ю—aluminium, Т—titanium, К—cobalt and Д—copper. The first two figures preceding the first letter indicate the average carbon content of the steel in hundredths of one per cent; the number following each letter indicates the content of the corresponding alloying element in per cent. No number is shown if the content is less than 1.5 per cent. The letter A at the end of the designation always indicates that the steel is of high quality. For example, alloy steel grade 35XH3MA is a high-quality steel containing 0.35% C, 1% Cr, 3% Ni and 1% Mo.

**Classification according to structural class.** On the basis of the structure obtained when specimens of small cross section are cooled in air, alloy steels may be classified as: (1) pearlitic, (2) martensitic, (3) austenitic, (4) ferritic and (5) carbidic, or ledeburitic.

Steel of the *pearlitic* (as well as sorbitic and troostitic) class is obtained when the amount of alloying elements is relatively small

(up to 5 per cent). It has good machinability and its mechanical properties are considerably improved by heat treatment (hardening and tempering).

At an alloying element content exceeding 5 per cent steel passes into the *martensitic* class. Such steels have a very high hardness and present difficulties in machining.

A very large content, from 10 to 30 per cent, of certain alloying elements (Ni, Mn or Co) enables the austenite structure to be retained in steel at room temperature; such steel belongs to the *austenitic* class. Stainless, nonmagnetic and heat-resistant steels, for example, are of the austenitic class.

Steels of the *ferritic* class contain a large amount of alloying elements (e. g., Cr, W or Si) but have a low carbon content. They do not respond to hardening; in the as-annealed condition their structure comprises alloyed ferrite and a small amount of cementite.

Steels of the *carbide* (ledeburitic) class contain considerable amounts of carbon and carbide-forming elements (Cr, W, Mn, Ti, Nb and Zr). Complex carbides formed in these steels are dispersed in the matrix which may be sorbitic, martensitic or austenitic, depending upon the chemical composition.

If alloy steel of the carbide class is forged, the carbide inclusions are uniformly distributed throughout the matrix in the form of separate globules. Carbide steels are chiefly used for making cutting tools.

Alloy, as well as carbon steels can be divided into hypoeutectoid, eutectoid and hypereutectoid types. As a rule, alloying elements shift the line *SE* of the iron-carbon equilibrium diagram to the left. Thus eutectoid alloy steel may be formed, not at 0.8 per cent but at a much lower carbon content.

**Classification according to purpose.** As to the uses for which their properties fit them, alloy steels can be classified as structural steels, tool steels and steels with special physical properties.

*Alloy structural steels* are divided into three groups depending upon the total content of alloying elements: low-alloy (up to 2 per cent alloying elements), medium-alloy (from 2 to 5 per cent) and high-alloy (over 5 per cent).

After suitable heat treatment alloy structural steels acquire higher mechanical properties than carbon structural steels. This is due to the higher hardenability and, consequently, more uniform properties along the cross section of alloy steels. This feature is of especial importance for parts with a large cross section.

Alloying elements introduced into structural steels strengthen the ferrite, refine the grain and increase the resistance to softening on heating to moderate temperatures.

Certain grades of alloy structural steels will be considered in more detail.

Low-alloy manganese structural steels contain from 0.9 to 1.8% Mn and up to 0.5% C. In steel, manganese forms solid solutions with ferrite and austenite, and carbides with carbon; it increases the hardenability of steel since it increases the stability of the austenite and lowers the critical point  $A_1$ .

A characteristic feature of manganese structural steels containing over 1.5% Mn is their susceptibility to temper brittleness; their impact strength is sharply reduced when they are slowly cooled after tempering at 400°-600° C.

Manganese steels with a low carbon content possess good plastic properties and respond easily to elongation and rolling in both hot and cold conditions.

Good weldability is one of the advantageous features of low-carbon manganese steels.

Manganese steel containing from 0.9 to 1% C and from 12 to 14% Mn acquires a purely austenitic structure after being quenched in water from 1000°-1050° C. It is used for parts requiring high toughness in conjunction with high wear resistance (buckets and lips of dredgers, excavators and draglines, stone-crusher jaws, railway track frogs and switches, etc.).

Chromium is not a critical alloying element in the U.S.S.R. and therefore chromium structural steels are extensively applied in industry.

The favourable effect of chromium is that it lowers the critical quenching rate and improves the hardenability of steel, increasing the stability of the austenite.

The tensile strength and yield point of chromium steels are increased by hardening and high-temperature tempering; their ductility is almost unaffected while the toughness is lowered somewhat.

The engineering industries use a great deal of chromium structural steels that contain from 0.15 to 0.5% C and from 0.7 to 1% Cr.

Also widely used is steel grade 15XA, containing 0.15% C and about 1% Cr. Parts made of this steel are subjected to carburising. To obtain a martensitic structure and a high hardness in the case, steel 15XA is quenched in oil instead of water, thereby reducing deformation of the part.

An addition of 0.1% V to chromium steel refines the structure of the metal. This has a favourable effect on the ductility of the steel, reduces its susceptibility to overheating and improves the mechanical properties, especially the impact strength.

Crankshafts and other critical machine parts are made of steel grade 40X. Steel 50XΦA is used in the manufacture of highly critical springs.

Chrome-molybdenum steel (36XMA) is used to make heavily loaded bolts, studs, gears and shafts. An addition of 0.4 to 0.55 per cent

molybdenum to steel improves its hardenability, reduces its susceptibility to overheating and eliminates temper brittleness. Its excellent weldability has found chrome-molybdenum steel applications in welded structures.

Silicon steels, grades 55C2 and 60C2A, are suitable for all possible types of springs; their mechanical properties after quenching and tempering are very high.

Chrome-silicon-manganese steel (chromansil type, grade 13XΓCA and others) has good welding properties. It is used to make shafts, axles and weldments that are to operate under alternating loads.

Nickel is a very important alloying element, but one that is in short supply. It should be employed only where other alloying elements are unsuitable. Nickel improves the hardenability of steel, increases the austenite stability and reduces the critical cooling rate.

The strength and hardness of nickel steels is increased by hardening without any appreciable reduction in ductility and toughness.

Straight nickel steels are seldom used. Much more extensively applied are complex steels in which nickel is used in conjunction with other alloying elements. Examples of such steels are the chrome-nickel grades which have good mechanical properties and high hardenability (to a depth of up to 100 mm). Steels 37XH3A and 40XH in particular are used for making heavily loaded parts of various cross sections; grades 12XH3A and 12XH4A—for gears, shafts, pins, axles and other parts undergoing carburising.

Another widely applied grade is chrome-nickel-tungsten steel (18XHBA); it is used after hardening and low-temperature tempering or after carburising followed by quenching and tempering.

*Alloy tool steels* are employed in tool manufacture in cases when the tool life provided by carbon steel is insufficient.

The chemical composition of an alloy tool steel depends upon a number of factors. Primary of these are the type of material to be machined, the cutting speed, chip thickness and the temperature developed on the cutting edges of the tool.

The tool industry is supplied with:

- (1) low-alloy steels which retain high hardness at temperatures up to 250° C; and
- (2) high-alloy steels which retain high hardness at temperatures up to 600° C.

Low-alloy tool steels, grades X, 9XC, XΓ, XBΓ and certain others, are of the eutectoid and hypereutectoid types. In comparison with carbon steels they have a somewhat higher hardness and wear resistance, i. e., the ability of retaining the cutting properties of the tool over a long period of operation.

High-alloy tool steels contain considerable quantities of tungsten, vanadium and chromium which are carbide-forming elements that

increase the ability of the steel to retain the hardness and cutting properties of the tool at elevated temperatures (red-hardness).

Most tools are manufactured of steel grade P18, containing 0.8% C, 18% W, 4% Cr and 1.5% V, and steel P9, containing 0.9% C, 9% W, 4% Cr and 2 to 2.5% V. These are known as high-speed steels because tools made of them can operate at high cutting speeds. They acquire high cutting properties only after suitable heat treatment.

A temperature vs time diagram indicating established practice for hardening and tempering high-speed steel, grade P18, is shown in Fig. 51. In hardening, this steel should be heated very slowly up to

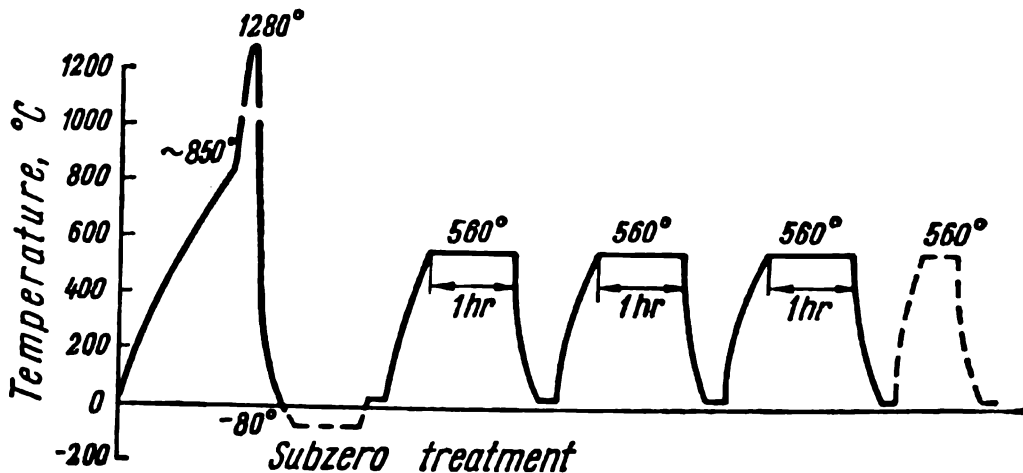


Fig. 51. Temperature vs time diagram for hardening and tempering high-speed steel, grade P18

850° C to avoid thermal stresses and cracking due to the low heat conductivity of the steel. Heating from 850° to 1280° C, when the steel is already in the plastic state, should be, on the contrary, very rapid so as to prevent decarburisation of the tool surfaces. High temperatures (1260°-1280° C) are required in hardening steel P18 to enable the secondary carbides to be dissolved in the austenite. After heating to the indicated temperatures the tools are quenched in oil.

A stepped quenching procedure in molten salts at temperatures from 500° to 550° C is widely applied for certain types of tools. This operation reduces the stresses and deformation in the metal.

As a result of hardening, high-speed steel acquires a structure consisting of martensite, retained austenite and complex carbides. The Rockwell hardness attained is within 61 to 63.

High-speed steel tools undergo high-temperature tempering at 560° C to obtain the specified properties after hardening. During this tempering operation carbides of chromium, tungsten and vanadium are precipitated from the retained austenite, increasing the hardness of the tool. Thus, a second hardening of the steel seems to take place. Single tempering of steel P18 does not ensure the trans-

formation of all the retained austenite into martensite and, therefore, multiple tempering is applied (usually three tempering operations of one hour holding time each). Subzero treatment of high-speed steel at  $80^{\circ}\text{C}$  below zero, directly following quenching, trans-

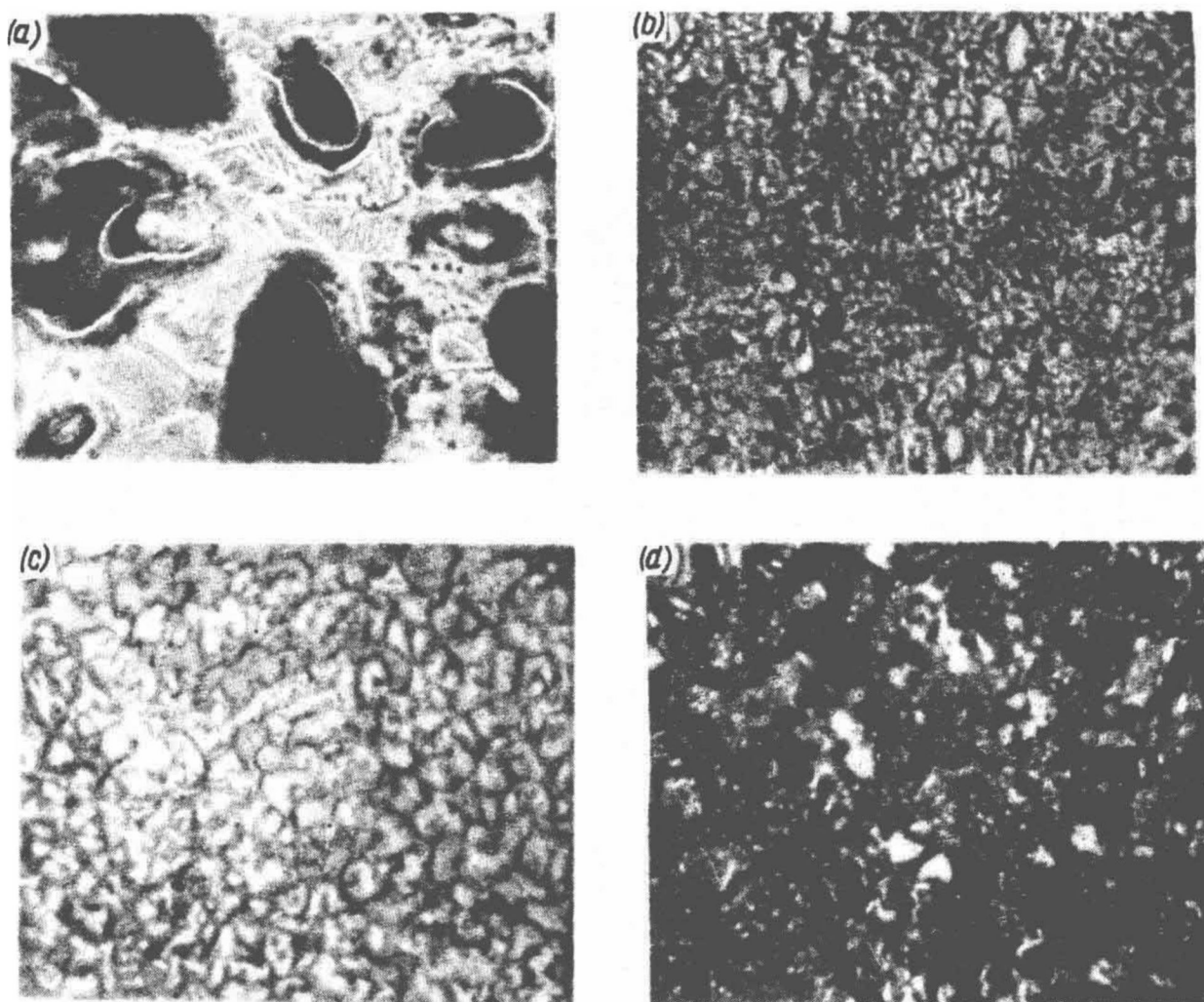


Fig. 52. Microstructure of high-speed steel P18:

(a) after casting, structure comprises pearlite+ledeburite+secondary carbides; (b) after forging and annealing, structure comprises sorbite-like pearlite+primary and secondary carbides; (c) after quenching from  $1280^{\circ}\text{C}$ , structure comprises austenite+primary carbides+martensite; (d) after quenching and triple tempering, structure comprises martensite+primary carbides

forms a considerable part of the retained austenite into martensite and enables tempering to be limited to two operations at  $560^{\circ}\text{C}$  for steel P18.

Fig. 52a illustrates the microstructure of high-speed steel, grade P18, after casting. The structure of the cast steel comprises pearlite, ledeburite and secondary carbides. The same steel, after forging and annealing, has a structure consisting of sorbitic pearlite and primary and secondary carbides (Fig. 52b). After quenching from a temperature of  $1280^{\circ}\text{C}$ , the steel acquires a structure of austenite, marten-

site and primary carbides (Fig. 52c). Triple tempering at 560° C gives the same steel a structure containing martensite and primary carbides as shown in Fig. 52d. Sometimes high-speed steel tools are subjected to low-temperature cyaniding after tempering to further improve their cutting properties.

*Alloy steels and alloys with special physical and chemical properties* can be divided into the following five groups: (1) stainless steels, (2) scale and heat resistant steels and alloys, (3) wear resistant steels, (4) magnetic steels and alloys, and (5) alloys with special thermal properties.

Stainless steel is distinguished by its high resistance to corrosion in various aggressive media—atmospheres of air or water vapour, sea water, solutions of acids or salts, etc.

The most extensively used stainless steels contain from 0.1 to 0.45% C and 12 to 14% Cr. Chromium forms a thin, dense film of chromium oxide on the surface of the steel which reliably protects the part against destruction in an aggressive medium.

High corrosion resistance is also offered by chrome-nickel steels of the austenitic class containing not over 0.12-0.14% C, from 17 to 20% Cr and 8 to 11% Ni.

Scale-resistant steels and alloys are capable of withstanding high temperatures for long periods without the formation of a layer of scale on their surfaces. This feature is achieved by the addition of chromium, aluminium and silicon which increase the resistance of steel to the attack of oxidising gases (gas corrosion) at high temperatures. The action of the aforesaid elements is based upon the formation of thin, dense oxide films on the surface of the steel or alloy for protecting the parent metal against scaling.

Chromium steels containing from 12 to 14% Cr are sufficiently scale-resistant up to 750°-800° C. An increase in the chromium content to 15-17 per cent extends the resistance to scaling to 850°-1000° C; at a content of 30% Cr, resistance is maintained up to 1100° C.

Complex alloy steels containing large amounts of silicon and chromium are known as silchrome steels (grades X6C, X9C2, X12IOC and X6CM). Their scale resistance remains high when they are heated to temperatures from 800° to 1100° C.

All kinds of scale-resistant steels and alloys are used in the manufacture of parts subject to the action of high temperatures but not that of heavy loads. In some cases scale-resistant steels are used to make heating elements of furnaces.

Heat-resistant steels and alloys should not only resist scaling at high temperatures but should retain their mechanical properties as well. This is achieved by a proper selection of the chemical composition and appropriate heat treatment of the steels. Parts subject to



high temperatures and pressures are frequently made of chrome-nickel-silicon steel, grade X25H20C2, which contains 23 to 27% Cr, 18 to 21% Ni and 2 to 3% Si. Parts of steam and gas turbines and valves are made of a more complex steel, grade 4X14H14B2M, whose composition includes 13 to 15% Cr, 13 to 15% Ni, 2 to 2.7% W, 0.25 to 0.4% Mo and 0.4 to 0.5% C.

Magnetic steels and alloys may be classified as permanent-magnet and soft-magnet types.

Soft-magnet steels and alloys are used in the production of pole cores, transformers, generators, motors and electromagnets (transformer steel and dynamo sheets). These steels and alloys must have a high magnetic permeability ( $\mu$ ) so that a high magnetic induction is produced in the core of an electromagnet by a relatively weak current passing through the winding. The coercive force ( $H_c$ ) must be low in soft-magnet materials.

The most commonly used soft-magnet material is sheet steel from 0.35 to 1 mm thick with a low carbon content (not over 0.1 per cent) and an increased silicon content (within 3.8 to 4.2% Si in transformer steel and within 0.8 to 2.3% Si in dynamo sheet).

Permanent-magnet steels and alloys are used, as the name implies, for making permanent magnets that are installed in many measuring instruments, radio apparatus, magnetos, etc. These materials must possess a high coercive force ( $H_c$ ) and a high residual induction ( $B_r$ ).

Permanent magnets are made of complex steels alloyed with Co, W, Cr, Ni, Cu and Al.

High magnetic properties in permanent-magnet steels and their stability are obtained not only by adding the necessary amount of alloying elements but by carefully observing heat treating techniques. These steels are hardened at high temperatures (near 1300° C) and are slowly cooled in a magnetic field. This is followed by a tempering operation at 600° C.

Alloys with a high nickel content (35 to 44 per cent) and a low carbon content (up to 0.35 per cent) have an austenitic structure which ensures a low linear coefficient of thermal expansion.

Three alloys with special thermal properties are used in the U.S.S.R.; they are: Invar (H36), Platinite (H42) and Elinvar (X8H36).

The coefficient of thermal expansion of *Invar* is near to zero in the temperature range from  $-50^\circ$  to  $+100^\circ$  C. At temperatures above 100° C, however, the coefficient rapidly increases and at 275° C exceeds that of ordinary steels. Invar is employed in precise instrument manufacture, e. g., for geodetic and optical instruments.

*Platinite* has a thermal expansion near to that of glass. It can easily be fused into glass and is therefore used as a substitute for expensive platinum.

*Elinvar* is an alloy with a modulus of elasticity that remains practically constant in the temperature interval from  $-50^{\circ}$  to  $+100^{\circ}$  C. It is used to make watch springs, tuning forks and various physical instruments.

### 10-3. Cutting Alloys

Cutting alloys are used on tools that operate at very high cutting speeds with high temperatures developing on the cutting edges (up to  $1000^{\circ}$ - $1100^{\circ}$  C). Since they cannot be machined by ordinary methods, they are applied to cutting tools either by means of hard-facing techniques (as a weld deposit) or as tips brazed to the tool shank or body.

As to the method by which they are produced, cutting alloys are classified as cast cutting alloys and cemented carbides.

*Cast cutting alloys*, or *stellites*, contain large amounts of the critical metals, cobalt and tungsten, have a high hardness ( $R_c=60-65$ ) and melt at very high temperatures. Tips or rods from 5 to 10 mm thick, cast of these alloys, are used in the hard-facing of tools by welding techniques to increase the life of the cutting edges. Dies and wear surfaces of various machine parts are hard-faced in the same manner.

Sormite and stalinite, stellite substitutes containing no cobalt or tungsten, have been developed in the Soviet Union. However, their tool life is inferior to that of stellites because of their high brittleness.

*Cemented carbides* are made of a powdery mixture of tungsten and titanium carbides and metallic cobalt which is first compacted and then sintered, thus employing the well-known powder-metallurgy process.

In their finished form cemented carbides consist of extremely fine grains of tungsten and titanium carbides with a cobalt binder. These alloys are extremely hard—with a Rockwell hardness number up to 85 and even higher—and they retain this hardness at temperatures up to  $1000^{\circ}$  C.

The most widely used Soviet grades of cemented carbide are BK2, BK3, BK6, BK10 and BK15, containing from 2 to 15% Co (the remainder is tungsten carbide), and T5K10, T14K8, T30K4 and T60K6, containing from 6 to 10% Co and 5, 14, 30 and 60 per cent titanium carbide, respectively (the remainder, again, being tungsten carbide).

The cutting properties of cemented carbides considerably exceed those of high-speed steel.

## Chapter 11

### NONFERROUS METALS AND ALLOYS

#### 11-1. Aluminium and Its Alloys

The most important properties of aluminium are its low specific gravity (2.7), high electrical and thermal conductivities, high ductility, and corrosion resistance in various media. Aluminium has a face-centred cubic crystal lattice whose constant depends upon its degree of purity. The mechanical properties of pure aluminium are not very high though it possesses good ductility.

Pure aluminium has only few applications; it is used for the manufacture of electrical wire, chemical apparatus, household utensils and for coating other metals.

More widely used in industry are aluminium alloys which can be divided into two main groups: (1) wrought alloys, from which a great variety of articles are made by means of rolling, forming, drawing, forging and pressworking and (2) casting alloys, which are employed in foundry practice.

Wrought aluminium alloys have a high mechanical strength which in some cases approaches that of steel. The ductility of these alloys, as well as their hot and cold workability, has found them extensive applications in all branches of industry.

Wrought aluminium alloys are further classified as (1) non-heat-treatable and (2) heat-treatable alloys.

Non-heat-treatable wrought aluminium alloys, i. e., those which do not respond to heat treatment, include the aluminium-manganese alloy, grade AM $\Pi$ , containing about 1.3% Mn, and the AM $\Gamma$  series of aluminium-magnesium-manganese alloys, containing about 2.5% Mg and 0.3% Mn.

Alloys, grades AM $\Pi$  and AM $\Gamma$ , possess sufficiently high mechanical strength and ductility, as well as good corrosion resistance and weldability; they are most frequently available in the form of sheets and special shapes.

The mechanical properties of the AM $\Pi$  and AM $\Gamma$  alloy series can be altered by strain-hardening. Thus, for example, for annealed AM $\Pi$ ,  $\sigma_b = 13$  kg/mm<sup>2</sup> and  $\delta = 20$  per cent while for the semistrain-hardened state,  $\sigma_b = 16$  kg/mm<sup>2</sup> and  $\delta = 10$  per cent.

Wrought aluminium alloys also include complex alloys of aluminium with copper, nickel, iron, silicon and other alloying elements.

When aluminium is alloyed with certain of these elements, systems with limited solid solubility or chemical compounds are formed,

improving the strength properties of the alloys and enabling heat treatment to be applied. This principle is clearly demonstrated by the binary aluminium-copper alloy system which is the basis of many engineering aluminium-base alloys. The part of the aluminium-copper equilibrium diagram shown in Fig. 53 includes the most important alloys in this system. It is evident from the diagram that aluminium and copper form the chemical compound  $\text{CuAl}_2$ , and solid solutions with a maximum copper solubility of 5.7 per cent at the eutectic temperature ( $548^\circ\text{C}$ ). Aluminium-copper alloys containing

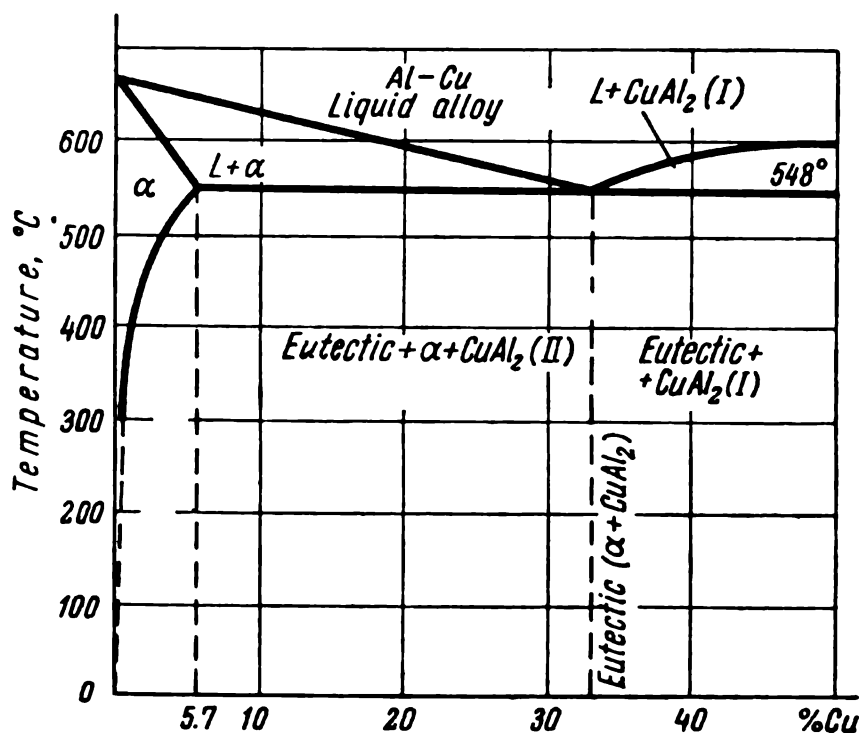


Fig. 53. Part of the aluminium-copper equilibrium diagram

less than 0.5% Cu acquire a single-phase structure after being slowly cooled (alpha solid solution of copper in aluminium). Alloys with a copper content from 0.5 to 5.7 per cent have a two-phase structure upon cooling under the same conditions (alpha solid solution in aluminium +  $\text{CuAl}_2$ ). If a two-phase alloy of aluminium and copper is heated to a temperature above the line of maximum solubility of copper in aluminium, the chemical compound  $\text{CuAl}_2$  dissolves in the aluminium and the structure of the alloy again becomes single-phased. This circumstance is used in the heat treatment of alloys of the Al-Cu system. This is called solution heat treatment and consists in heating the alloy to the single-phase region, quenching (rapid cooling) and subsequent artificial ageing (prolonged holding at  $100^\circ\text{C}$ - $150^\circ\text{C}$ ) of the hardened alloy.

In the solution heat treatment of aluminium-copper alloys the chemical compound  $\text{CuAl}_2$  does not have time to precipitate from

the alpha solid solution which is fixed in the unstable (supersaturated) state. The return of the supersaturated solid solution to the stable state can be easily observed in the artificial ageing of solution heat treated aluminium-copper alloys.

During artificial ageing extremely fine (dispersed) particles of  $\text{CuAl}_2$  are precipitated, thus strengthening the alloy.

If the alloys are heated to higher temperatures or held for a long time at a temperature of  $150^\circ\text{C}$ , coagulation of the fine  $\text{CuAl}_2$  particles is observed and the alloys lose the additional strength acquired in heat treatment.

The mechanical properties of heat-treatable binary alloys of the aluminium-copper system can be improved by adding other alloying elements.

Complex wrought aluminium alloys of the duralumin (dural) type and certain others have found most extensive application in many industries.

Duralumin (dural) belongs to the Al-Cu-Mg system to which manganese has been added (to increase the strength and corrosion resistance of the metal), and which includes silicon and iron as inevitable minor constituents.

Typical duraluminium alloys contain from 2.2 to 5.2% Cu, up to 1.75% Mg, up to 1% Si, up to 1% Fe and up to 1% Mn. The main hardening phase in the heat treatment of duralumin consists of the dispersed particles of the ternary compound  $\text{CuMgAl}_2$ .

Several grades of duralumin are available in the U.S.S.R. They are identified by the Russian letter Д followed by a figure indicating the number of the alloy in the series. Duralumin, grade Д-1, can be obtained in the form of sheets, bar stock and tubing; grades Д-6 and Д-16 are usually produced in the form of bars, and grade Д-3П is made as wire for rivets.

The heat treatment of duralumin involves:

- (1) annealing at  $360^\circ\text{C}$ ;
- (2) quenching in water; grade Д-1 is heated before quenching to  $505^\circ\text{--}510^\circ\text{C}$  and grades Д-6 and Д-16 to  $495^\circ\text{--}503^\circ\text{C}$ ;
- (3) natural or artificial ageing.

Natural ageing, carried out at room temperature for several days, imparts high mechanical properties to the metal and good corrosion resistance in various media. Elevated temperatures, i. e., artificial ageing, accelerate the process but duralumin does not acquire its maximum mechanical properties and its corrosion resistance is lowered to some extent.

*Duralumin* (except grade Д-3П) in the as-annealed condition has a tensile strength  $\sigma_b = 19$  to  $21\text{ kg/mm}^2$ , elongation  $\delta = 18$  per cent and hardness  $\text{Bhn} = 42\text{--}45$ . These properties change substantially after solution heat treatment and natural ageing (with the exception of

grade Д-3П): tensile strength increases to 42-44 kg/mm<sup>2</sup>, and elongation drops to 15-17 per cent. If duralumin is heated to 260° C and held for a short time (20 to 40 sec) after hardening, the initial properties of the alloy are restored. This phenomenon is called *recovery*. It increases the ductility of the metal and is applied in practice in cold working operations.

**Aluminium forging alloys.** Aluminium alloys used in smith and closed-die forging processes are designated as grades AK2, AK4, AK6 and AK8 (the letters being the abbreviation of the Russian for "aluminium, forging"). According to the grade, these complex alloys contain from 1.9 to 5% Cu, 0.4 to 1.8% Mg, up to 0.2% Mn, 0.5 to 1.2% Si and 1 to 2.3% Ni.

Forgings of the AK series of aluminium alloys are heated to 510°-520° C, quenched in water and then artificially aged for 12 to 18 hours at 150°-160° C.

These alloys are used in the manufacture of pistons (AK2); vanes, screws, housings, motors, and pump impellers (AK6), as well as rings, covers, disks and other parts.

**Aluminium-base Casting Alloys.** Aluminium casting alloys are identified by the letters АЛ (abbreviation of the Russian for "aluminium, casting") and a figure which is simply the ordinal number of the alloy in the series.

The principal elements added to these aluminium-base alloys are silicon, copper, magnesium and zinc. Aluminium alloys containing from 8 to 14% Si are called *silumin alloys*. Alloys АЛ2 (with from 10 to 13% Si and 0.8% Cu) and АЛ4 (with from 8 to 10% Si, 0.3% Mg and up to 0.5% Mn) possess good casting properties, and ample ductility and corrosion resistance.

Solution heat treatment and ageing have a very small effect on the mechanical properties of silumin alloys. Their strength, however, can be increased without any heat treatment if they are modified in the molten condition with a small amount of sodium.

Modification of molten silumin alloys is usually done by a ladle addition of either metallic sodium in an amount equal to 1 per cent of the melt by weight or a mixture of  $\frac{2}{3}$  sodium fluoride and  $\frac{1}{3}$  sodium chloride, the amount being 2 per cent by weight. The change in the microstructure of a silumin alloy due to modification can be seen in Fig. 54. The photomicrograph of the modified alloy shows the light-coloured primarily precipitated aluminium and the fine-grained eutectic with thin globules of silicon. Such a structure is associated with an improvement in the mechanical properties of the silumin alloy; the tensile strength  $\sigma_b$  is increased from 12-16 to 18-22 kg/mm<sup>2</sup> and the elongation  $\delta$ , from 1 to 3-10 per cent.

Grades АЛ2 and АЛ4 are used both for sand and permanent-mould castings.

Substantial changes can be effected in the strength of silumin alloys by adding certain amounts of manganese, copper and zinc.

The aluminium-base casting alloys, grades АЛ6 (with an addition of 2 to 3% Cu), АЛ8 (7 to 14% Cu) and АЛ11 (10 to 14% Zn), belong to the group of special silumin alloys. The castability of these alloys,

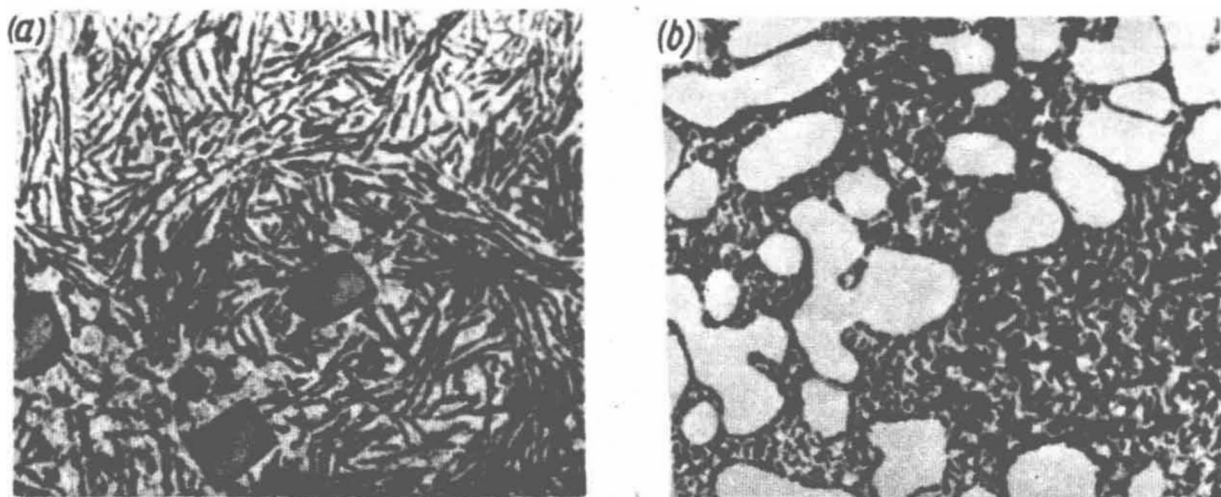


Fig. 54. Microstructure of a silumin alloy:  
(a) before modification; (b) after modification

with the exception of АЛ11, is inferior to that of simple silumin alloys АЛ2 and АЛ4, but they respond to hardening by solution heat treatment followed by ageing.

## 11-2. Magnesium and Its Alloys

Magnesium has a specific gravity of approximately 1.7; its alloys are the lightest of all engineering metals employed.

The melting point of magnesium is  $650^{\circ}\text{C}$ ; its boiling point is  $1007^{\circ}\text{C}$ . Magnesium is very inflammable and burns with a dazzling flame, developing a great deal of heat.

The mechanical properties of magnesium, especially the tensile strength, are very low and therefore pure magnesium is not employed in engineering.

The alloys of magnesium possess much better mechanical properties which ensure their sufficiently wide application.

The principal alloying elements in magnesium alloys are aluminium, zinc and manganese. Aluminium, added in amounts up to 11 per cent, increases the hardness, tensile strength and fluidity of the alloy. Up to 2 per cent zinc is added to improve the ductility (relative elongation) and castability. The addition of 0.1-0.5 per cent manganese raises the corrosion resistance of magnesium alloys.

Small additions of cerium, zirconium and beryllium enable a fine-grained structure to be obtained; they also increase the ductility and oxidation resistance of the alloys at elevated temperatures.

Magnesium alloys are classified into two groups:

(1) Wrought alloys, grades MA1 (up to 0.3% Al and 1.3-2.5% Mn) and MA2 (up to 3-4% Al, 0.6% Zn and 0.5% Mn).

(2) Casting alloys, grades MЛ4 (5-7% Al, 2-3% Zn and up to 0.5% Mn) and MЛ5 (8% Al, 0.6% Zn and up to 0.5% Mn).

*Wrought magnesium alloys* MA1 and MA2 are chiefly used for hot smith and closed-die forged machine parts. They are less frequently used as sheets, tubing or bar stock. The tensile strength of these alloys  $\sigma_b = 21$  to  $26 \text{ kg/mm}^2$  and their elongation  $\delta = 8$  per cent.

*Magnesium casting alloys* MЛ4 and MЛ5 are widely used as foundry material though their castability is inferior to that of aluminium-base alloys.

The mechanical properties of magnesium casting alloys can be considerably increased by heat treatment. This consists in heating the alloy to  $380^\circ \text{C}$  (MЛ4) or  $415^\circ \text{C}$  (MЛ5), cooling in air and ageing for 15 or 16 hours at a temperature of  $175^\circ \text{C}$ . This treatment can increase the tensile strength  $\sigma_b$  from 16-17 to  $25 \text{ kg/mm}^2$ .

### 11-3. Copper and Its Alloys

**Copper.** Copper is a valuable metal. Its wide application in many fields of engineering is due to its exceptionally high electrical and thermal conductivity, low oxidisability, good ductility ( $\delta$ —up to 50 per cent) and to the fact that it is the basis of the important industrial alloys, bronze and brass.

Various grades of copper are used for engineering purposes; in the U.S.S.R. their chemical composition is specified by the state standards. It must be noted that even a minute amount of impurities sharply alters the properties of pure copper. For example, commercial copper containing 0.013% O, 0.24% Al and 0.55% Sb has only one-third the electrical conductivity of pure copper. Lead, bismuth and certain other elements have no substantial effect on the electrical conductivity but reduce the ductility to a great degree.

The mechanical strength of pure copper is not high and depends upon the degree of deformation (reduction in working).

Pure copper is used chiefly for electrical engineering products such as cables, busbars and wire.

More widely employed are the copper alloys. The alloying of copper with other elements increases the strength of the metal in some cases and improves the anticorrosive and antifriction properties in others.

Copper-base alloys comprise two main groups—brasses and bronzes.





The two-phase alloys, (alpha + beta) brasses, are chiefly used in the form of tubing, sheets, bar stock of various shapes and other semifabricated products. Single-phase (alpha) and two-phase (alpha + beta) brasses have good workability (undergo plastic deformation easily) in both the hot and cold conditions when they are subjected to intermediate annealing operations.

If care is taken in the working of brass to keep the grain size in the metal within 0.05 mm, high mechanical properties are obtained. Admixtures of bismuth and lead detrimentally affect the hot workability of alpha brass. This is due to the formation of fusible eutectics arranged on the grain boundaries and leading to red-shortness.

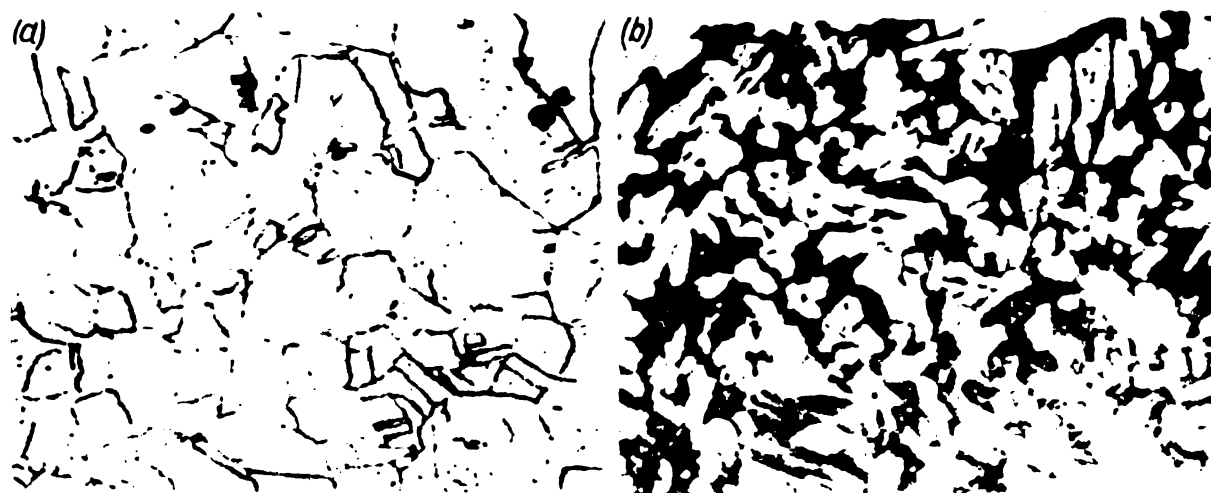


Fig. 56. Microstructures of brasses:  
(a) alpha brass; (b) (alpha+beta) brass

In the U.S.S.R. brasses are identified by means of the Russian letter Л (the first letter of the Russian word for brass) followed by letters designating the chief elements contained and numbers indicating the mean percentage content of these elements. Thus, grade ЛТ96 is the brass tombac (T) containing 96% Cu and Zn. The designation of grade ЛМЖЦ-59-1-1 indicates that the brass contains 59% Cu, 1% Fe, 1% Mn, the remainder being Zn.

Special brasses containing aluminium, manganese, nickel, silicon and other elements have properties greatly differing from those of ordinary brasses. In particular, manganese (ЛМЦ-58-2) and iron-manganese (ЛМЖЦ-59-1-1) brasses are characterised by their high mechanical strength ( $\sigma_b$  = up to 65 kg/mm<sup>2</sup>); nickel brass (ЛН-65-5) possesses increased corrosion resistance and aluminium-nickel brass (АЛН-59-3-2) has high strength in conjunction with an excellent corrosion resistance.

**Bronzes.** Alloys of copper with a number of elements including tin, aluminium, silicon, manganese, iron and beryllium are called *bronzes*.

Fig. 57 shows the equilibrium diagram for the copper-tin system which is the basis of ordinary bronzes. Only the left side of the diagram, limited by a tin content of 25-30 per cent, has practical application in industry.

It can be seen from the diagram that slowly cooled copper-tin alloys containing up to 14% Sn consist of grains of the alpha solid solution of tin in copper (single-phase bronzes). The mechanical properties of ordinary bronzes in the alpha-phase region vary with the tin content in the following manner. The hardness and strength of the alloy increase with the tin content. The ductility increases up to a tin content of 5 per cent and then falls sharply. At a tin content in excess of 14 per cent bronze becomes double-phased. The structure then comprises areas of the eutectoid ( $\alpha + \text{Cu}_3\text{Sn}_8$ ) in addition to the grains of solid solution.

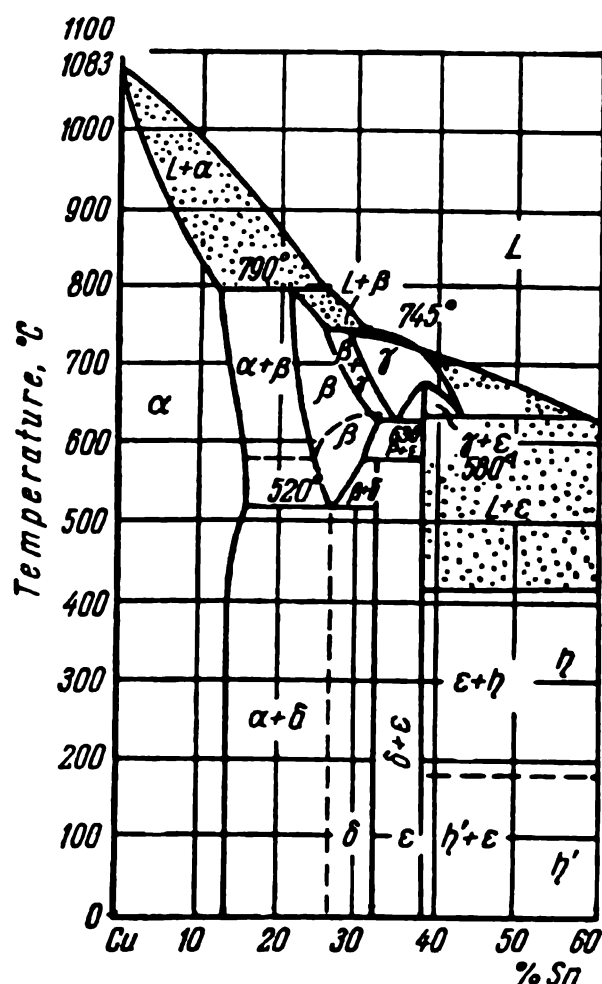


Fig. 57. Part of the copper-tin equilibrium diagram

The structure of bronze with 7% Sn is illustrated in Fig. 58. It consists of the alpha solid solution of tin in copper and the eutectoid ( $\alpha + \delta$ ).

The presence of the chemical compound  $\text{Cu}_3\text{Sn}_8$  in bronzes makes them harder and more brittle.

Tin bronzes are divided into two groups: wrought bronzes, containing up to 6% Sn, and casting bronzes, containing over 6% Sn.

*Special bronzes* are copper-base alloys in which the principal admixtures are Al, Ni, Mn, Si, Fe, Be and others.

Special bronzes are fully equivalent substitutes for the more expensive tin bronzes and, therefore, of great economical value.

These bronzes are designated on the same principle as brasses. The designation begins with the Russian letters *Бр* (the first two letters of the Russian for bronze) which are followed by letters indicating the main elements and numbers showing the average percentage of these elements.

Certain grades of special bronzes deserve more detailed consideration.

Aluminium bronzes contain from 4 to 11% Al; their high mechanical properties and corrosion resistance considerably surpass those of tin bronzes and brasses.

The castability of aluminium bronzes is good and they are frequently used in foundry practice. Sheets, strips, bars and wire are made of grades БрА5 and БрА4 by the rolling process. Aluminium bronzes with admixtures of iron and manganese, grades БрАЖ9-4, БрАЖМЦ10-3-1.5 and БрАМЦ9-2, are suitable for castings and for working, especially for smith and closed-die forging. Bronzes with

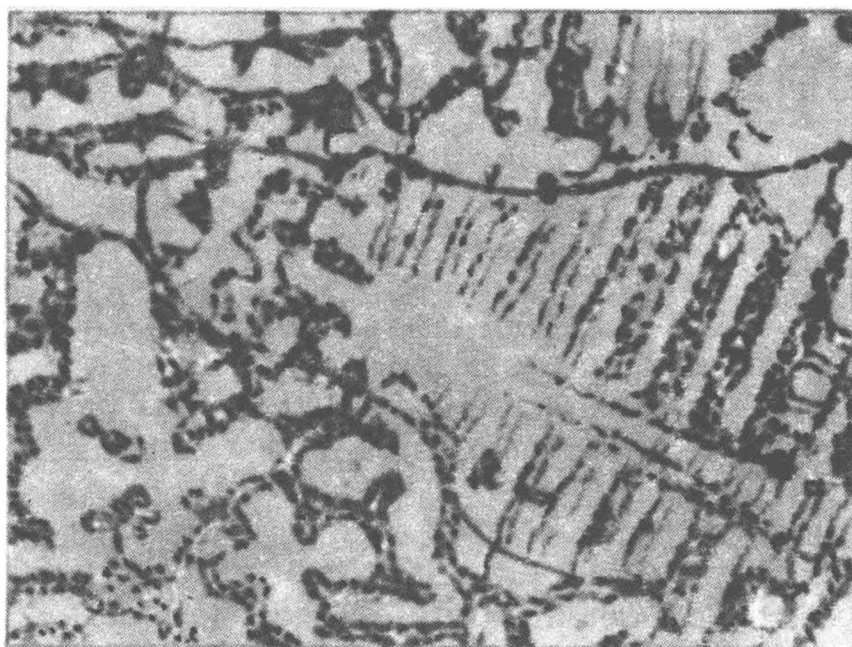


Fig. 58. Microstructure of as-cast tin bronze (7% Sn)

additions of iron and nickel, grades БрАЖН10-4-4 and БрАЖН11-6-6, possess high mechanical strength, wear resistance and ample heat resistance; they are used for furnace fittings and machine parts with frictional surfaces.

Silicon bronzes present a fortunate combination of high mechanical, antifriction and anticorrosive properties with good casting qualities and weldability. Additions of manganese, nickel, zinc and lead enable silicon bronzes to be employed as substitutes for tin bronzes in the most critical applications.

Beryllium bronzes are characterised by high mechanical strength, corrosion resistance, antifriction qualities and good ductility. Other advantageous features are their high electrical and thermal conductivity.

Up to 2 per cent beryllium is added to bronze. Its solubility in copper at a temperature of 860° C is 2.8 per cent while at room temperature the solubility is reduced to 0.2 per cent. This circumstance

enables beryllium bronzes to be effectively precipitation hardened. The hardened alloy (quenched from  $800^{\circ}\text{C}$ ) undergoes artificial ageing at  $250^{\circ}\text{C}$ - $350^{\circ}\text{C}$  after which it acquires high strength and elasticity.

Beryllium bronzes are used for making the springs of electrical contacts and watch movements, as well as diaphragms and other elastic parts.

#### 11-4. Antifriction Alloys

Antifriction alloys are used mainly for lining (babbitting) bearings and, for this reason, they are also called *bearing metals*.

A bearing metal must satisfy a number of requirements. The coefficient of friction between the journal and lining must be as low as possible; the metal must be such as to prevent excessive wear or heating of the friction surfaces. At the same time a bearing metal must withstand high specific loads, retain sufficient strength and hardness when heated to elevated temperatures, be capable of being rapidly run in with the shaft and promote the formation of an oil film on the lining surface when the bearing is lubricated.

Modern bearing metals may be classified into babbits, antifriction bronzes, antifriction cast irons and nonmetallic antifriction materials.

*Babbitts* are used for casting the linings of bearings since they are capable of reducing the friction and wear of friction surfaces. Babbitts should not be harder than the metal of the shaft journal as otherwise they may scratch and score the mating surface.

Babbitts consist of tin, antimony, lead and copper; other elements possessing good antifriction qualities are added in some cases.

The base of babbits used in industry is lead or other metals capable of substituting for more expensive tin.

The standard designation for babbits is a letter and number system in which the Russian letter Б stands for babbitt while the number directly following the letter Б indicates the percentage of the mean tin content. If the babbitt contains special admixtures they are indicated by a second letter following the letter Б. Thus, the grade identification, Б83, of the high-tin babbitt indicates that it contains 83% Sn; grade БН is a babbitt to which nickel has been added and grade БТ is one with tellurium.

The high-tin babbits, grades Б83 and Б89, are the best bearing metals. They provide for a low coefficient of friction both with and without lubricant, possess ample strength and begin to melt only at  $240^{\circ}$  or  $250^{\circ}\text{C}$ . They find applications in critical machinery.

The new standard alloys БН (with nickel) and БТ (with tellurium) and others can be employed as substitutes for the high-tin babbitt Б83 in many plain bearing applications.

A calcium babbitt is used as the standard bearing metal for railway rolling stock. Besides lead it contains 0.8 to 1.1% Ca and 0.75 to 1.0% Na.

Antifriction (bearing) bronzes are used in plain bearings and other mechanisms with friction surfaces subject to a high specific load. Tin-lead bronzes, containing from 4 to 16 per cent lead, are used for this purpose. Lately, from 2 to 7% Zn is being added to tin-lead bronzes. Straight tin and other types of antifriction bronzes are also extensively used.

Antifriction cast irons are suitable for plain bearings operating under high specific pressures but at low shaft speeds. High-quality grey cast irons, grades C4U and C4U2, are used as materials for the liners or shells of bearings and other parts subject to friction. They contain from 3.2 to 3.6% C, 2.2 to 2.4% Si and 0.6 to 0.9% Mn, and have a pearlitic structure with a normal amount of graphite inclusions in the metallic matrix. These cast irons have a hardness of Bhn = 170-229.

Recently various nonmetallic antifriction materials have been developed for industrial applications. They include laminated fabric, viozite, plasticised wood and others.

### 11-5. Titanium and Its Alloys

As an engineering material titanium has been widely applied only in the last years.

Titanium is a silvery-white metal which melts at approximately 1668° C and has a specific gravity of 4.505.

There are two allotropic modifications of titanium, namely: alpha-titanium, stable at temperatures up to 885° C, and beta-titanium, stable at higher temperatures. Commercially pure titanium possesses high strength properties:  $\sigma_b = 66 \text{ kg/mm}^2$ ,  $\delta = 35 \text{ per cent}$  and Bhn = 230. The tensile strength of most titanium alloys ranges from 100 to 140 kg/mm<sup>2</sup>, in conjunction with high elongation.

The hardness, tensile strength and yield point of titanium are increased with the degree of cold deformation to which it is subjected. The elongation value drops rapidly when the degree of cold deformation (reduction) exceeds 50 per cent and becomes equal to 10 per cent.

Impurities found in commercial titanium can be divided into two groups: elements which form interstitial solid solutions with titanium (O<sub>2</sub>, N, C and H<sub>2</sub>) and elements which form substitutional solid solutions (Fe and other metallic elements). The first have a much greater effect on the mechanical properties than those in the second group.

Even very small amounts of oxygen and nitrogen in titanium alloys sharply reduce the ductility. A carbon content of more than 0.2

per cent reduces both the ductility and impact strength of a titanium alloy.

It is supposed that the brittleness of titanium is a result of strain ageing and is connected with the presence of dissolved hydrogen in the beta-phase. An oxidising atmosphere, free of water vapour, should be used whenever possible in the heat treatment of titanium.

Titanium and its alloys are hardened either by a surface heat treatment followed by ageing at 400°-540° C or by producing a case containing nitrogen, carbon and boron, using an appropriate procedure.

Industrial titanium alloys contain vanadium, molybdenum, chromium, manganese, aluminium, tin, iron or other elements, singly or in various combinations. These alloys respond to heat treatment, casehardening and work hardening techniques.

A combination of high mechanical properties with a low specific weight and excellent corrosion resistance enables titanium to be used in building supersonic aircraft.

## Chapter 12

### CORROSION OF METALS AND ALLOYS

#### 12-1. General Concepts

Almost all metals and alloys subject to the action of atmospheric air or other surrounding media (for example, sea water, soil, acid and alkali solutions, organic liquids, etc.) are gradually destroyed, beginning from the surface, and lose their initial appearance. This progressive destruction of a metallic surface exposed to an external aggressive (active) medium is called *corrosion*.

Experience shows that corrosive destruction depends mainly upon the following three factors: (1) the chemical nature of the metal or composition of the alloy and their structures; (2) the chemical nature of the surrounding medium and the percentage of aggressive matter it contains (oxygen, moisture, acids, alkalis, etc.) and (3) the temperature of the surrounding medium.

As to its character, metal corrosion may be classified as: (1) *uniform corrosion*, in which the whole surface of the metal or alloy is corroded with equal intensiveness; (2) *localised corrosion*, in which only certain areas of the surface are attacked; (3) *selective corrosion*, where only separate structural components of an alloy are affected and (4) *intercrystalline corrosion*, which involves destruction of the metal or alloy along its grain boundaries.

According to the mechanism of the corrosion process, it is necessary to distinguish between chemical and electrochemical corrosion.

*Chemical corrosion* conforms to the laws of chemical kinetics; it may be observed in its pure form only if no moisture condenses on the surface of the metal. A typical example of chemical corrosion is the oxidation and erosion of the valves of internal combustion engines by the incandescent products of combustion. This type also covers metal corrosion in nonelectrolytes of an organic nature, such as sulphur-bearing crude oils and masout.

A film of corrosion products, usually oxides, is formed on the surface of metal in the course of chemical destruction. In some cases this film may protect the underlying metal against further corrosion, i. e., make it more passive in respect to the surrounding medium. Such protection is ensured only if the passivating film is continuous. This becomes possible if the volume of metal oxide obtained in the course of corrosion,  $V_{ox}$ , exceeds the volume of oxidised metal,  $V_{met}$ , i. e.,  $V_{ox} > V_{met}$ .

Comparatively dense oxide films, capable of protecting metal against further oxidation, are formed on the surface of aluminium, lead, tin, nickel and chromium. Friable films are produced in the oxidation of barium, calcium and magnesium. A sufficiently dense film is formed if iron is oxidised in dry air or in an atmosphere of oxygen, but it begins to crack and scale off the metal as it becomes thicker.

*Electrochemical corrosion* occurs in the presence of liquids which are electrolytes, i. e., ones containing free ions. The essence of electrochemical corrosion is that the atoms, on the surface of the metal in contact with the electrolytic solution, pass into the solution as ions and leave an equivalent quantity of electrons in the metal.

The tendency of the atoms of a given metal to enter a liquid electrolyte as ions determines the electrochemical potential of the metal. The electrochemical potential of a metal indicates the voltage of the electric current, or electromotive force, that must be applied to the interface between the solid metal and liquid electrolyte to prevent the metal from sending its ions into the solution. The electrochemical potential is a value proportional to the energy required for one metal ion to pass into the solution under the given conditions. Table 3 is what is called an electromotive force series and indicates the electrochemical potentials for the case when certain metals dissolve in a normal solution of the sulphate of the given metal (normal electrochemical potentials).

The more negative the normal electrochemical potential of a metal is, the more pronounced its tendency to dissolve in electrolytes (aque-



Table 3

Electromotive Force Series for Various Elements

Metal	Normal electrochemical potential, V	Metal	Normal electrochemical potential, V
Copper . . . .	+0.33	Cadmium . . .	−0.40
Bismuth . . .	+0.28	Iron . . . . .	−0.44
Antimony . . .	+0.22	Chromium . . .	−0.56
Tin . . . . .	−0.10	Zinc . . . . .	−0.76
Lead . . . . .	−0.12	Manganese . . .	−1.10
Nickel . . . . .	−0.23	Aluminium . . .	−1.34
Cobalt . . . . .	−0.29	Magnesium . . .	−1.53

ous solutions of salts, acids and alkalis). Thus, for example, the dissolution of lead, i. e., its corrosion in a diluted solution of sulphuric acid, is much more difficult than the dissolution of iron, etc.

Another aspect of electrolytic corrosion must be mentioned. If two dissimilar metals are in contact and are exposed to (or immersed in) a corrosive electrolyte, they form a so-called galvanic couple with a definite difference in potentials.

The formation of a galvanic couple in the given system leads to the development of spontaneous dissolution of the metal possessing the more negative electrode potential. This metal acts as the anode and its ions enter the electrolytic solution. As far as the second metal, with the less pronounced negative electrode potential, is concerned, it becomes the cathode and does not dissolve as long as the first metal is present in the electrolyte. For example, if iron and zinc are the contacting couple and are immersed in the electrolyte (e. g., in a weak solution of sulphuric acid), zinc dissolves more rapidly than iron and protects the latter against corrosion (the electrode potential of zinc is more negative than that of iron).

In corrosion processes galvanic couples may be formed, not only between separate pieces of contacting metals with equal electrode potentials, but also between microscopic crystals of a single alloy if the crystals differ in composition and physical properties.

When a structurally inhomogeneous alloy is exposed to an electrolyte, microscopic anodic and cathodic areas appear on the surface of the alloy. Microscopic or even submicroscopic galvanic cells formed by these areas lead to corrosive destruction of the alloy. In many cases, destruction occurs due to a selective attack in which corrosion penetrates deep into the metal along the grain boundaries (intercrystalline or intergranular corrosion).

## 12-2. Methods for Combating Corrosion

Prevention of corrosion consists in the selection of the proper composition and structure of metals and alloys in product design, those that are most resistant to the particular corroding medium, and of the use of rational storage and handling procedures. Corrosion prevention measures are of prime importance.

The susceptibility of a metal or alloy to corrosion in a given aggressive medium is determined primarily by its chemical nature and degree of structural homogeneity. Corrosion of a metal is intensified if it contains microscopic areas (even grains) which differ in chemical composition and/or structure so that galvanic microcells are formed. Therefore, a metal with a homogeneous structure offers better resistance to corrosion than one comprised of several phases.

Sound product design is of great significance in corrosion prevention. The shape of a part should be selected in such a way as to prevent the collection of moisture in slots, corners, joints, etc. Moreover, all mating parts or articles should be made of materials near to each other in the electrochemical series.

The principal corrosion prevention and protection methods applied in practice are:

(1) Alloying metals to obtain chemically inactive alloys of special composition.

(2) Forming oxide films on the surfaces of metal parts.

(3) Applying protective metallic coatings on the parts.

(4) Protecting the surface of metal with a coat of paint or lacquer.

*Additions of alloying elements* to the composition of certain metals (especially steel and cast iron) increase their corrosion resistance in many cases.

The introduction of a large and sometimes of a small amount of alloying elements may prevent or reduce corrosion. In the first case the alloying element content may reach 20 or 30 per cent (stainless steels). Alloying additions always form solid solutions with the parent metal (austenitic steels, aluminium bronzes, etc.) and this substantially increases the corrosion resistance of the alloy. For example, small additions of copper and chromium (less than 1 per cent) increase the corrosion resistance of steel.

Alloying elements added in a small amount take part in forming a protective (passivating) oxide film or they have a considerable effect on the electrode potentials of the structural constituents of the alloy.

*Passive oxide films* are produced artificially on certain metals and alloys by treatment in an acid or alkaline medium in the presence of strong oxidising agents (anodic oxidation). It is frequently necessary to coat an easily corroded metal or alloy with another metal to insulate the part from the aggressive environment.

*Corrosion protection by metallic coatings* is extensively practised at the present time.

Metallic coatings are of two types—anodic and cathodic.

Metals used in anodic coatings have a more negative electrode potential (high solution pressure) than the parent metal to be protected. The anodic coating is dissolved in the course of corrosion thereby protecting the parent metal against destruction. Scratches or breaks in anodic coatings do not reduce their protective ability since such breaks in continuity of the coating lead chiefly to destruction of the coating itself.

Metals used for cathodic coatings, on the contrary, must have a lower solution pressure, i. e., their electrode potential must have a less negative value. As long as the cathodic coating completely prevents contact between the part and the corroding medium, no corrosion of the part occurs. Breaks in the thin coating, due to scratches or mechanical damage, end its protective action in reference to the parent metal since the latter is more easily destroyed when exposed to an electrolyte.

Zinc and cadmium serve as anodic coatings for ferrous alloys; copper, tin, lead, nickel and others are used in cathodic coating.

All protective metallic coatings must be continuous; no scratches, dents or other defects are permitted on the surfaces of parts to be coated.

Metallic coatings may be applied by hot-dip methods, by electroplating and by spraying with a Schoop gun.

Coating with molten tin is used in the manufacture of tinplate (tins for preserved foods, tinning kettles for cooking food, etc.). Zinc dipping or hot-dip galvanising is used for coating roofing iron, water pipe and other outdoor hardware. Hot-dip lead coating protects chemical apparatus against corrosion.

In electroplating (electrodeposition) the metal part is immersed and held in a plating tank where an electric current deposits the plating metal on the surface of the part. Nickel, chromium, zinc and other types of electroplating are widely employed.

Metal spraying (metallising) is done by blowing fine droplets of molten metal, by means of a blast of compressed air, on the surface to be coated using a spray gun or pistol.

The purpose of *coating metal parts with a layer of paint or lacquer* is to prevent moisture from coming into contact with the surface of the metal. Such treatment is compulsory for all steel structures (sheets, trusses, columns, etc.). Several types of lacquers and paints, however, are used to protect metals against electrochemical corrosion as well.

The use of *protectors*, or *sacrificial anodes*, is often resorted to in protecting metals. This method is based on the fact that when two dissimilar metals are immersed in an electrolyte, the metal with the

lower electrode potential becomes the anode and is gradually destroyed, thereby protecting the other metal, which is the cathode, against corrosion. This principle is used, for instance, to protect the steel hulls of sea-going vessels by hanging zinc protector sheets around them.

Pipelines, boilers and other steam power facilities through which water flows are corroded and thereby acquire a negative electric charge. Corrosion can be stopped by producing a positive charge. This is done by electrically connecting the installation to the negative pole of a d-c generator to apply an impressed current after which the pipes, boilers, etc., become cathodes in reference to the negative charge of the generator and cease to corrode.

## FUNDAMENTALS OF METAL CASTING

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### *Chapter 13*

#### MOULDING PROCESSES

Many metallic articles may be produced in the foundry in the form of castings. This method involves melting the metal and then pouring it into a previously made mould where it solidifies as a casting whose shape and size are a reproduction of the mould cavity.

Castings are made of grey and malleable irons, steels and copper-, aluminium- and magnesium-base alloys.

Foundry processes enable parts of intricate shape to be obtained, e. g., cylinder blocks of automobile and airplane engines, pistons and piston rings, machine tool beds and frames, mill rolls, wheels and housings of steam and hydraulic turbines, water supply and sewer pipes, etc. This feature has found castings the widest of applications in the engineering, metallurgical, building, chemical and other industries.

A complicated sequence of operations is required in metal casting. The main processing operations in making a casting are shown by the flow diagram in Fig. 59.

#### **13-1. Patterns, Moulding Tools and Flasks**

**Patterns.** The most common method of making castings is in sand moulds which are made in metallic flasks using wooden or metal patterns. Moulds may also be made of metal (permanent moulds) or of fireclay (loan moulds).

The pattern is not an exact copy of the part to be cast since it incorporates projections (core prints) which form recesses in the sand

to hold cores in the mould (cores, inserted into moulds to form through or blind cavities, are made separately).

The pattern and mould required to produce a simple casting—a short cast-iron pipe with two flanges—are illustrated in Fig. 60.

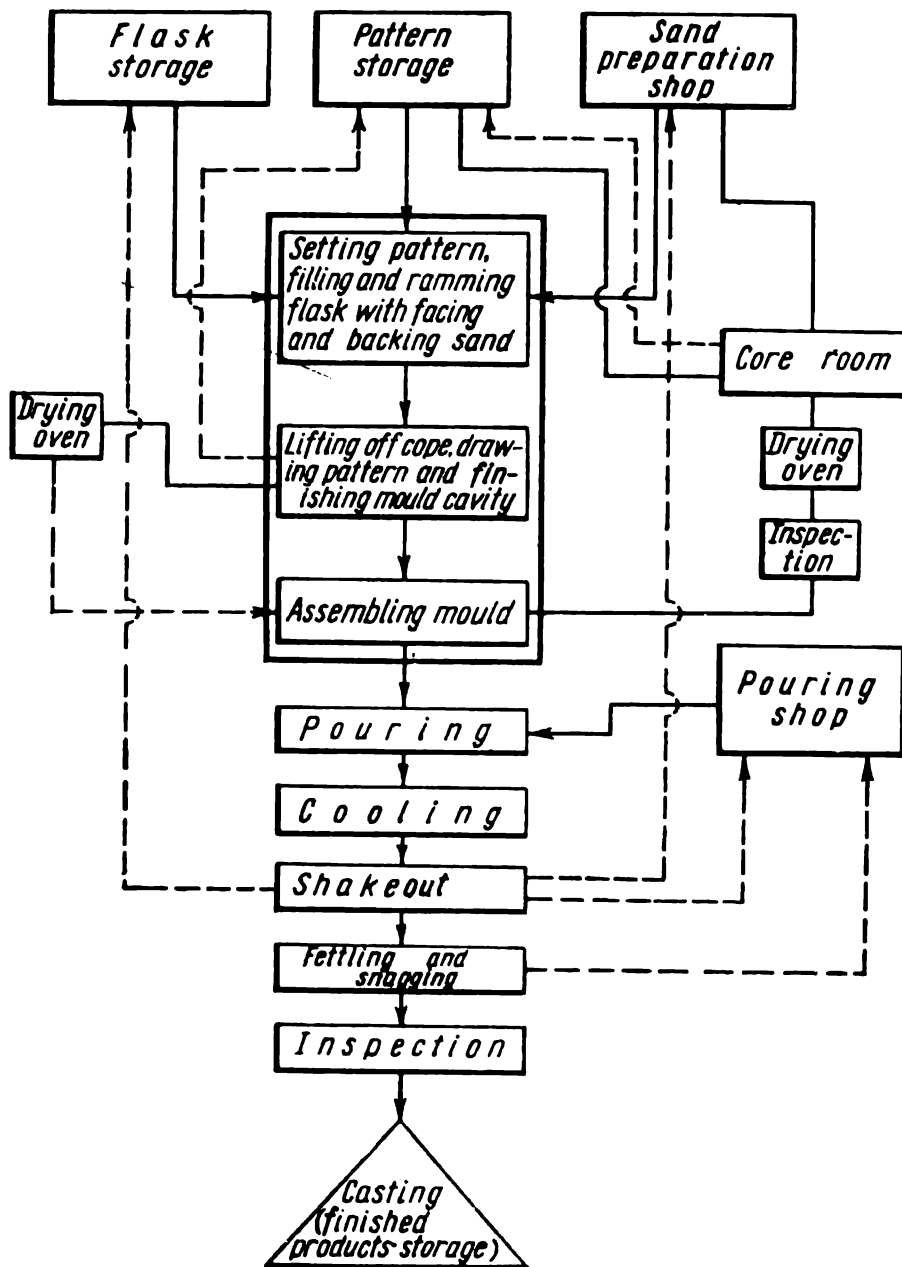


Fig. 59. Flow chart of processing operations in making a casting

The pattern to which the parts are cast, the core box by means of which the cores are made, mould boards, and the pattern plates on which the patterns are secured in moulding constitute *the pattern equipment*.

The pattern equipment may be either of wood, metal or of a combination of the two.

Wooden patterns must be made of dried or seasoned lumber (containing not more than 10 per cent moisture) to avoid warping and distortion in subsequent drying. The working surface of a pattern should be smooth and wear resistant. Patterns should not absorb moisture from the moulding sand or from the surrounding air. This is prevented by coating them with paint and waterproof varnish.

The following colour code is accepted in Soviet foundries for indicating the kind of metal to be cast: red—for grey iron, blue—for steel, and yellow—for nonferrous castings. Core prints on patterns are painted black and loose parts of patterns are edged with a black stripe.

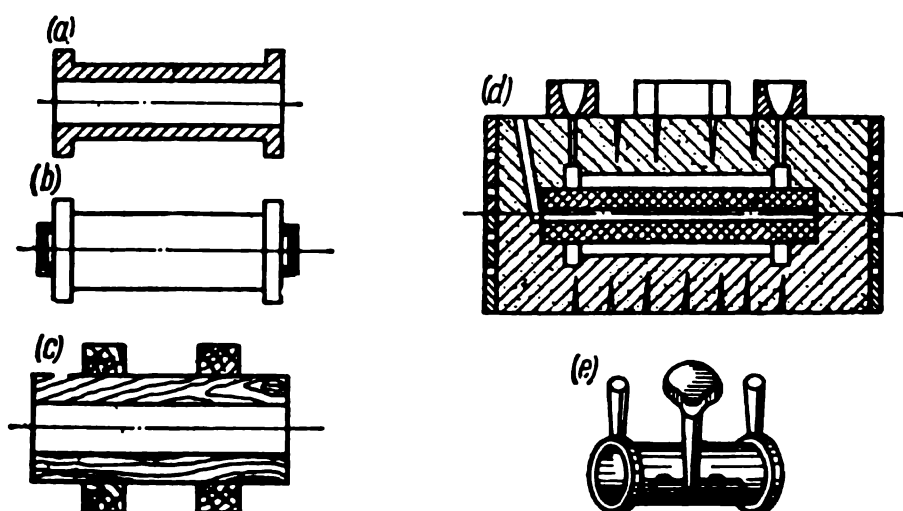


Fig. 60. Moulding a cast-iron pipe:

(a) finished casting; (b) wooden pattern; (c) wooden core box; (d) assembled sand mould in a two-piece flask; (e) casting with the gating system and risers

Since metals shrink in solidification and contract in further cooling to room temperature, linear dimensions of patterns are increased in respect to those of the finished casting to be obtained. This is called the shrinkage allowance and amounts to from 1 to 2 per cent for grey iron castings, 2 to 3 per cent for steel and 1 to 2 per cent for nonferrous castings depending upon the composition of the alloy.

Draft is the taper allowed on the vertical faces of a pattern or core box cavity to permit the pattern to be removed from the mould or the core from the box. It ranges from  $1^\circ$  to  $3^\circ$  for wooden patterns and equals  $5^\circ$  for core boxes.

Patterns for parts that are to be subsequently machined provide for additional stock on the corresponding surfaces. This is called the machine-finish allowance.

Patterns are designed so as to avoid sharp and abrupt changes in section at the junctions of parts of the casting. External and internal corners of patterns are suitably rounded; they are called *rounded corners* and *fillets*, respectively.

Metallic patterns are most often made of aluminium-base alloys which are light in weight and can easily be machined.

**Moulding tools.** The tools used in sand moulding may be classified into two groups:

(1) Tools used for filling the flask with sand and for compacting the sand. These include shovels, hand and pneumatic rammers and other devices.

(2) Tools used for drawing the pattern from the mould and for finishing the mould surfaces, such as lifters, rapping bars, draw screws, draw spikes, trowels, scoops, slicks, spray cans, etc.

**Flask.** The rigid frame in which the moulding material is rammed to form the mould is called the *flask*. The purpose of the flask is to impart the necessary rigidity and strength to the sand in moulding and, in the case of permanent flasks, in conveying and pouring the mould. Flasks are made of steel, cast iron or aluminium alloys.

### 13-2. Initial Materials, Properties and Types of Moulding and Core Sands

**Initial (raw) materials.** The principal ingredient of a moulding sand, or mixture, is *silica sand*. Silica sand contains from 94 to 97%  $\text{SiO}_2$ , and is characterised by a high softening temperature and thermal stability.

In addition to silica sand a certain amount of *refractory clay* is required in sand moulding. Clay is the binder, imparting plasticity to the moulding sand in the moist state and increasing its strength after drying.

According to the amount of clayey matter they contain, moulding sands are classified as: silica (up to 2 per cent clay); lean (weak) (2 to 10 per cent clay); moderately strong (10 to 20 per cent clay); strong (up to 30 per cent clay) and extra strong sand (up to 50 per cent clay).

Clays used in moulding, containing finely dispersed mineral particles of aluminosilicates (e. g., kaolinite  $\text{Al}_2\text{O}_3 \cdot 2\text{SiO}_2 \cdot \text{H}_2\text{O}$  and others) are divided, according to their mineralogical composition, into ordinary and bentonite clays.

Additives, such as wood flour, sea coal (coal dust), peat and others, to moulding and core sands impart to them special properties and enhance others.

**Main properties of moulding sands.** Moulding sands are usually prepared by mixing fresh silica sand and refractory clay with a definite amount of used sand and various additives.

Moulding sands must have sufficient gas permeability, strength, plasticity, collapsibility and refractoriness. These properties are determined, not only by the chemical composition, but by the amount



of clayey matter in the sand, by its moisture content and by the shape and size of the silica sand grains.

Molten metal always contains a certain amount of dissolved gases which are evolved when the metal freezes. If these gases and water vapours evolved by the moulding sand do not have the opportunity to escape completely through the walls of the mould they may penetrate the liquid metal where, after solidification, they form gas holes and pores. Rejects due to the presence of gas holes in the castings are avoided if the moulding sand allows gases to pass through the whole thickness of the mould and escape, i. e., if the moulding sand has good *gas permeability*.

Permeability is expressed in conventional units which characterise the ability of the moulding sand to pass a definite volume of gases in unit time under standard conditions. Thus, the permeability of moulding sand should range from 30 to 80 conventional units for casting grey iron; from 70 to 100 for casting steel and should be about 30 for casting bronze, brass and aluminium. Core sands should have higher permeability than moulding sands; their permeability in the green (unbaked) condition should reach 70 to 130 units. Baked cores have even higher permeability.

Ample *strength* is another important property of moulding sands. Insufficient strength may lead to a collapse in the mould or its partial destruction during conveying, turning over or closing. The mould may also be damaged during pouring by washing of the walls and core by the molten metal. The sand and clay particles may be washed into the casting where they form sand inclusions (sand holes). When the mould is filled with molten metal, its bottom and walls are subjected to a high metalostatic pressure. If the sand is not strong enough to resist this pressure the mould cavity is enlarged and the castings obtained are oversize.

High plasticity is required of a moulding sand to obtain a good impression of the pattern in the mould.

*Plasticity* of moulding sand refers to its ability to acquire a pre-determined shape under pressure and to retain this shape when the pressure is removed.

A certain degree of *collapsibility* is required from moulding sands. This means their ability to decrease in volume to some extent under the compressive forces developed by the shrinkage of the metal during freezing and subsequent cooling. This property is especially important for cores. Lack of collapsibility in the moulding sand may result in the formation of cracks in the casting.

The grains of moulding sand must not melt, soften or sinter under the action of the high temperatures developed when molten metal is poured into the mould. This property is called *refractoriness*. Moulding sands with a poor refractoriness may burn onto the casting.

The chemical composition of the initial materials making up a moulding sand affects its refractoriness. The presence of the oxides of alkali or alkali-earth metals ( $\text{Na}_2\text{O}$ ,  $\text{K}_2\text{O}$ ,  $\text{MgO}$  and  $\text{CaO}$ ), for example, lowers the refractoriness of silica sand. Therefore, their content in the finished moulding sand should not exceed 1.5 to 2 per cent for steel castings, 5 per cent for heavy grey-iron castings and 7 per cent for medium and small grey-iron castings. No more than 3 to 4 per cent ferric oxide ( $\text{Fe}_2\text{O}_3$ ) may be allowed in moulding sands.

Optimum properties of moulding sand depend upon a proper moisture content which is expressed as a percentage. Moulding sands exhibit maximum strength at a moisture content of 4 per cent for lean sands and of 6 to 7 per cent for loam sands.

The granular constitution of sands, i. e., the shape and size of the quartz grains, has a substantial effect on the processing properties of moulding and core sands. The shape of the grains and number of similar grains in the sand determine the possibility of its application in various types of foundry practice.

**Types of moulding sand.** *Facing sand* is used directly next to the surface of the pattern and comes into contact with the molten metal when the mould is poured. Consequently, it is subjected to the severest conditions and must possess, therefore, high strength and refractoriness. It is made of silica sand and clay, without the addition of used sand.

The layer of facing sand in a mould usually ranges from 20 to 30 mm. From 10 to 15 per cent of the whole amount of moulding sand used in the foundry is facing sand.

*Backing sand* is used to back up the facing sand and to fill the whole volume of the flask. Old, repeatedly used moulding sand is mainly employed for this purpose.

In mechanised foundries where machine moulding is employed a so-called *system sand* is used to fill the whole flask. Since the whole mould is made of this system sand the strength, permeability, and refractoriness of the sand must be higher than those of backing sand.

Facing sand is always used to make dry-sand moulds while system sand is frequently employed for green-sand moulding.

Grey-iron castings are produced both in green- and dry-sand moulds. As a measure of economy a considerable amount of used sand (from 50 to 90 per cent) is added to moulding sands for grey-iron castings. Moreover, from 10 to 50 per cent fresh materials—sand and clay—and from 2 to 8 per cent pulverised coal are added to the mixture. The amount of fresh materials and coal added depends upon the weight and size of the casting; the heavier the casting and the thicker its walls, the more fresh materials are required.

Steel castings weighing up to 500 kg can be made in green-sand moulds; heavier castings require dry-sand moulds. If the moulding sand for steel castings is prepared of only fresh materials then it should comprise from 88 to 90 per cent silica sand and 10 to 12 per cent refractory clay for green-sand moulding and from 85 to 88 per cent silica sand and 12 to 15 per cent refractory clay for dry-sand moulding.

Moulding sands for nonferrous castings contain a considerable amount of clay and incorporate sands of finer grain. The melting point of nonferrous metals is much lower than that of ferrous metals and therefore less refractoriness and gas permeability is required of these moulding sands. A smooth surface is desirable on nonferrous castings; this is obtained by employing fine-grained sands.

**Core sands.** Cores are subjected to extremely severe conditions during the pouring of a mould since they are surrounded from all sides with molten metal. Gases can escape only through a small part of the core that is not environed by metal (the core print) and consequently have a long way to go, sometimes through the whole length of the core.

Special vent holes are provided in the core to assist in gas escape but this proves insufficient; good permeability is required of the core sand itself. Core oils, dextrin, resins and other special binding materials are used in core sands.

Core sands must also have a high refractoriness.

**Sand additives and facings to prevent burn-on and metal penetration.** Various miscellaneous materials (additives)—pulverised coal, graphite, peat, wood flour and other organic matter—are added to fresh sands and clays to impart special properties to moulding and core sands. Upon contact with the molten metal such additives burn and form gases which do not allow the metal to come into intimate contact with the sand grains. This gas “jacket” not only prevents interaction between the metal of the casting and the mould material but also makes the mould more collapsible when the metal shrinks.

Pulverised coal, graphite and charcoal are used as additives to moulding sands to prevent burn-on and metal penetration in case of grey-iron castings. These materials are finely ground and applied to the mould surfaces in the form of a dust coating (blacking).

Peat and wood flour (2 to 3 per cent) are added in dry-sand moulding to the moulding sand and also to core sands with a clay binder to improve their plasticity and collapsibility.

Dry-sand moulds are coated with whitening which has a high refractoriness. Whitening eliminates the possibility of burn-on and enables castings with smooth surfaces to be obtained. Whitening for grey-iron castings consists chiefly of graphite; that for steel castings, of silica flour.

### **13-3. Sand Preparation for Moulding and Coremaking**

Foundries use a great deal of moulding and core sands. From 4 to 5 cu m of moulding sand is expended to make one ton of sound castings.

**Preparing moulding materials for mixing.** Preparing the fresh materials involves drying, grinding, mulling and riddling.

New sand and clay contain a considerable amount of moisture and are therefore dried before use. The silica sand employed should have a moisture content not in excess of 6 per cent, otherwise it must be dried.

Loam sand used to renovate the sand circulating in the foundry should not contain more than 8 per cent moisture.

Sand is dried in either vertical or horizontal ovens. As a rule clay is dried only in horizontal ovens.

Grinding the raw materials, such as coal, clay, used cores, etc., is one of the important operations in the process of preparing moulding materials.

Jaw, roll and swing hammer crushers are employed in foundries for coarse grinding and edge runner and ball mills for fine grinding. Sand preparation systems, widely used in large foundries, have two edge runner mills, one for grinding coal and the other for clay.

After the clay and coal are ground to powder they are conveyed by a fan through a pipeline to closed hoppers (cyclones) where their speed is reduced and they settle to the bottom. From the cyclones the finished product is transferred to the sand preparation department by a pneumatic conveyer.

Clay is usually added to moulding and core sands in the form of a dry powder. From the standpoint of plant sanitation and employees' health it is desirable to add the clay in the form of a suspension in water.

The most effective devices for grinding moulding materials are vibration mills which operate on the principle of small pulses and a high frequency of repeated impacts.

**Reconditioning the circulating (used) sand.** The circulating (used) moulding sand shaken out of the flasks must be reconditioned before being used again.

In mechanised sand handling systems the used sand travels from the shake-out screens on a belt conveyer to the sand preparation department where large lumps are crushed, metal particles are removed and the sand is screened.

Metal particles are removed by magnetic separators installed at places where the used sand is transferred from one conveyer to the next.

**Sand preparation.** Before mixing all moulding materials are screened to remove large lumps of clay, pebbles, metal particles and other

foreign matter which may lead to casting spoilage. Shaker, rotary and vibratory screens are used for this purpose.

Mixing of moulding materials should ensure uniform distribution of the clay, moisture and other constituents between the sand grains. The more uniform this distribution, the better the main qualities of the sand—its vapour and gas permeability, and strength with the least amount of binding material.

Mixers used for sand mixing and conditioning may be of either the batch or continuous type.

It is easier to maintain a constant proportion of the sand components and moisture content in batch-type mixers since all the materials are charged simultaneously. On the other hand, these mixers have a lower output than the continuous type and consume more electric energy.

A continuous screw mixer is illustrated in Fig. 61. It comprises a trough in which the material is mixed by two screws or worms. The screws are arranged so that upon rotation of their shafts, they not only mix the material in the trough but gradually convey it from one end of the trough to the other. In the first half of the trough the materials are mixed dry. A moistener in the form of a water pipe is mounted above the second half of the trough; it sprinkles the sand with fine streams of water. The moistened sand is thoroughly mixed in travelling to the end of the trough where it is discharged through the outlet. Mixers of this type manufactured in the U.S.S.R. have capacities up to 60 cu m per hour.

A disadvantage of screw mixers is that they produce moulding sand of lower quality than that prepared in edge runner mixers (sand mullers) due to insufficient mixing and rubbing of the quartz grains.

Sand mullers (Fig. 62) have pan capacities of 0.3 or 0.6 cu m. If this type of muller is equipped with mechanised charging facilities including a special proportioner and operates continuously it can prepare 15 batches of system sand per hour, i. e., 4.5 cu m.

Sand mullers consist of a stationary circular pan 1 in which two plain muller wheels 2 and 3 rotate about a stationary vertical shaft 4. The wheels, in turn, rotate about horizontal shafts as they roll over the sand. Thus the wheels rotate both about their axes and around the pan. The wheels of mullers differ from the rolls of crushers in that the former run at a certain distance above the bottom of the pan. This leads to mixing and not crushing of the quartz grains in the sand. The muller wheels can lift slightly to run over large lumps of hard material.

The moulding sand is lined up under the muller wheels by special plows 5 and 6 which rotate together with the wheels about vertical shaft 4. Plow 5 pushes the sand under wheel 2 and plow 6 under wheel 3.

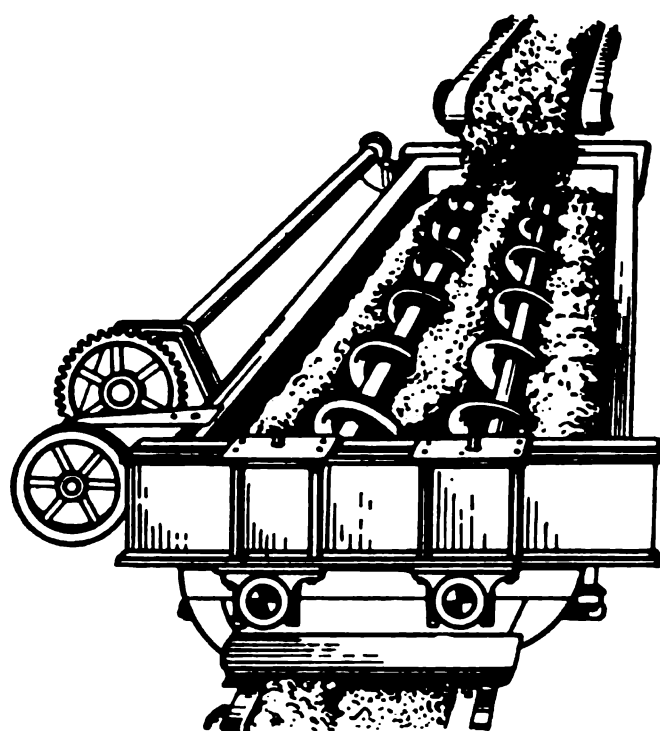


Fig. 61. Continuous screw-type sand mixer

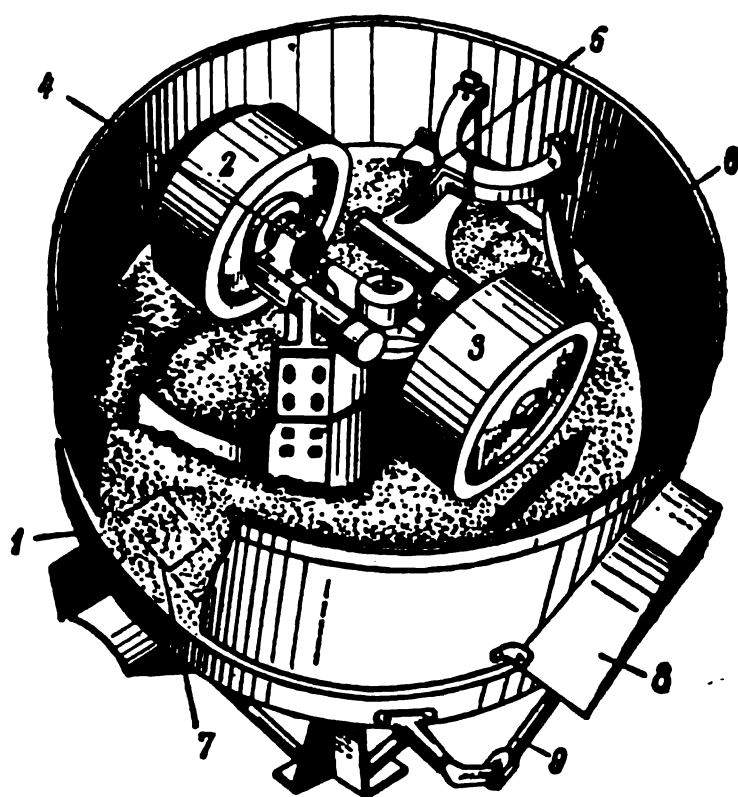


Fig. 62. Sand muller

The finished sand mixture is discharged by the plows through hatches 7 in the bottom of the pan while the wheels continue to roll around the pan. These hatches are closed during mixing. The opening and closing of hatches is mechanised by means of pneumatic cylinders, mounted in housing 8, which operate the hatches through tie rods 9.

In recent years foundries with a large output and requiring a correspondingly large amount of moulding sand have been employing more productive centrifugal-type speed muller sand mixers with an output up to 50 tons per hour.

The moulding sand obtained in the sand muller requires aeration to separate the sand grains. This operation, sometimes called fluffing, is performed by aerators of various design.

### 13-4. Moulding Processes

In piece and small-lot production foundry practice sand moulds are made by hand; moulding machines are employed in large-lot and mass production.

**Moulding procedures.** The most extensively used types of hand moulding are floor moulding in flasks or pit moulding to patterns and sweep moulding.

In *pit moulding*, all the work in making the mould is done on the earth floor of the foundry. Pit moulds may be either open or covered. In open pit moulding the upper part of the mould in the pit remains open; in the covered type the top is finished off with cores or with sand rammed in a cope flask.

Pit moulding requires that the earth floor at the moulding site be horizontal and sufficiently permeable to gases. Therefore, the place must be properly prepared beforehand. This is called "making the mould bed".

A bed of sand is prepared for smaller castings. The making of an occasional moulding pit for large and high castings involves covering the rammed bottom of the pit with a 50 to 80 mm layer of coke to improve the permeability of the mould. Vent pipes are run from this layer to the surface (at the sides), and the coke is covered with backing sand.

*Open pit moulding* (Fig. 63a) is used in casting parts of simple shape in which the upper surface is flat (plates, grate bars, pads, etc.). Pattern 1 is placed face downwards on the sand bed and is sunk carefully into the sand with light hammer blows applied through a board placed on the pattern. The horizontal position of the pattern is checked with the spirit level 4. Next the pattern is covered at the sides with facing sand which is then compacted, and backing sand is added. The sand is rammed first with a peen and then with a butt (flat) rammer. After this the pattern is checked again with the

level, excess sand is struck off, the sand around the pattern is smoothed with a trowel and vent holes 3 are pierced with a vent wire.

To pour the mould, a runner 6 is cut at one side and above it a pouring basin 5 is placed. From pouring basin 5 molten metal passes along runner 6 into the mould cavity. Runner 7 with a basin is provided at the other side of the mould to drain off excess metal.

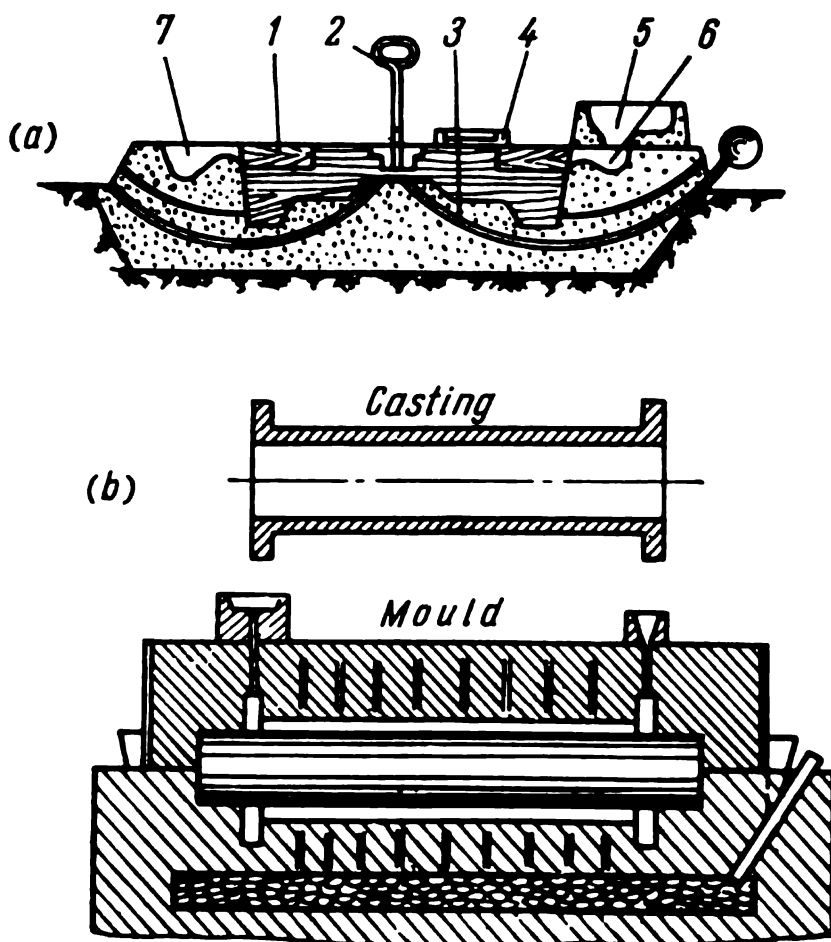


Fig. 63. Pit moulding:  
(a) open pit mould; (b) covered pit mould

When all the preceding operations have been performed, the pattern is drawn with draw screw 2. The impression of the pattern, i. e., the mould cavity, remains in the sand.

Parts of the mould which were broken off in drawing the pattern are repaired and smoothed down. Then the mould surfaces are dusted with graphite and the metal is poured. Immediately after pouring, the surface of the casting is covered with fine charcoal and a layer of dry sand to ensure uniform cooling of the casting and to prevent oxidation of the metal.

*Covered pit moulding* is done in an occasional moulding pit; it enables castings of more complex shape to be obtained with smooth clean surfaces.



An example of covered pit moulding is illustrated in Fig. 63b. Here the lower part of the pattern is placed in the previously prepared pit and is bedded into the sand to a certain depth. Next all the operations of open pit moulding are performed and then the upper part of the pattern is assembled to the lower part. The surface of the mould around the pattern is covered with fine dry parting sand and an empty cope is placed over the pattern. Wooden pegs locate the cope properly in respect to the lower part of the mould.

The patterns for the gate on the left flange and the flow-off on the right flange are set before covering the pattern with facing sand. Then the cope is filled with backing sand, rammed and vented. The moulded cope is lifted off with a crane, turned upside down and placed to one side. After this the lower half of the pattern is drawn from the pit and the top half from the cope. The surface of the mould cavity is repaired wherever necessary and finished, the separately made core is set into the mould, the cope is assembled to the lower part of the mould using the pegs again for location and a pouring basin is put on the mould above the gate.

Pit moulds are frequently dried before pouring. This may be done by means of a portable mould drier. The cope may be dried in a special drying oven.

Pit moulding is practised only in piece or job production.

*Flask moulding* is the most widely used process in both hand and machine moulding practice.

Various flask moulding methods may be applied depending on the shape, size and complexity of the casting to be made. These are: (1) two-part moulding with an unsplit pattern, (2) two-part moulding with a split pattern, (3) multiple-part moulding, (4) moulding with patterns having loose pieces, (5) stack moulding, (6) snap or removable flask moulding, etc.

A simple two-part mould using an unsplit pattern (Fig. 64) is made as follows. First the pattern is placed with its flat surface on mould board 1. Drag 2 (bottom part of the flask) is set over the pattern on the same board. After powdering the pattern with lycopodium, talc or graphite, a 15 to 20 mm layer of facing sand 3 is riddled over the pattern. The drag is then filled by layers of backing sand 4 from 70 to 100 mm thick, compacting each layer with a hand or pneumatic rammer 5 or by means of a machine. The top of the mould is rammed with the butt end of rammer 6.

After filling the drag, the excess sand is struck off with a wooden or steel straightedge 7 to make a level surface, and small vent holes 8 are pierced with a vent wire. These holes pass through the whole thickness of the mould, thus improving its permeability. The vent holes should not reach the pattern by 15 to 20 mm as otherwise they may spoil the mould and, moreover, metal may run into them during pouring.

Then the drag is turned upside down with the parting plane 9 upwards and is put on a levelled layer of earth. The surface of the mould is repaired wherever required and smoothed with a trowel. It is then covered with a fine coating of dry parting sand 10, removing excess

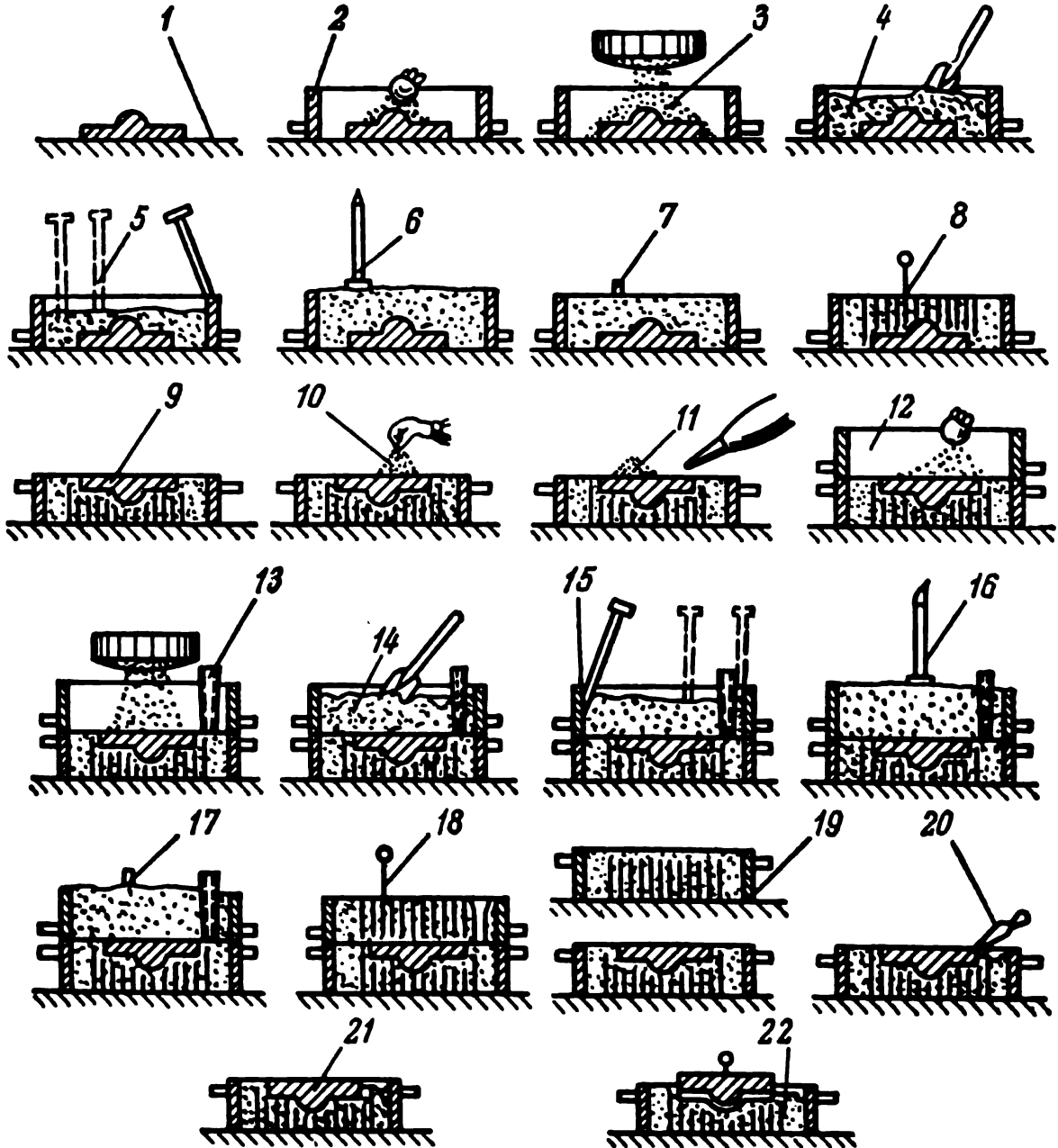


Fig. 64. Procedure for moulding an unsplit pattern in a two-part mould

sand 11 with hand bellows. The cope 12 (top part of the flask) is fitted on the drag and the pattern is powdered.

A sprue pin 13 is placed to one side of the pattern which is riddled over with facing sand and then the cope is filled with backing sand 14. The sand is compacted by ramming (15 and 16), levelled off with straightedge 17, and vent holes 18 are made.

Following these operations the tapered sprue pin is pulled out of the cope and a funnel-shaped opening is scooped out at the top of the sprue to form the pouring basin. Next the cope 19 is carefully lifted off and placed on a board with the parting plane upwards. The runner and gate 20 are cut in the drag, from the pattern to the sprue.

The sand around the edges of pattern 21 is moistened slightly with a swab to prevent the edges of the mould from crumbling when the pattern is drawn.

The pattern is drawn from mould 22 by means of either a draw screw or spike.

Dirt remaining in the mould cavity is blown out with a stream of air. Both halves of the mould are dusted with graphite or coal to reduce burn-on of sand on the casting surfaces.

Finally, the mould is assembled, the cope being carefully placed on the drag so that the flask pins fit into the bushes. A weight is placed on the top of the assembled mould or the halves are fastened together with shackles or bolts to prevent the molten metal from separating the cope and drag.

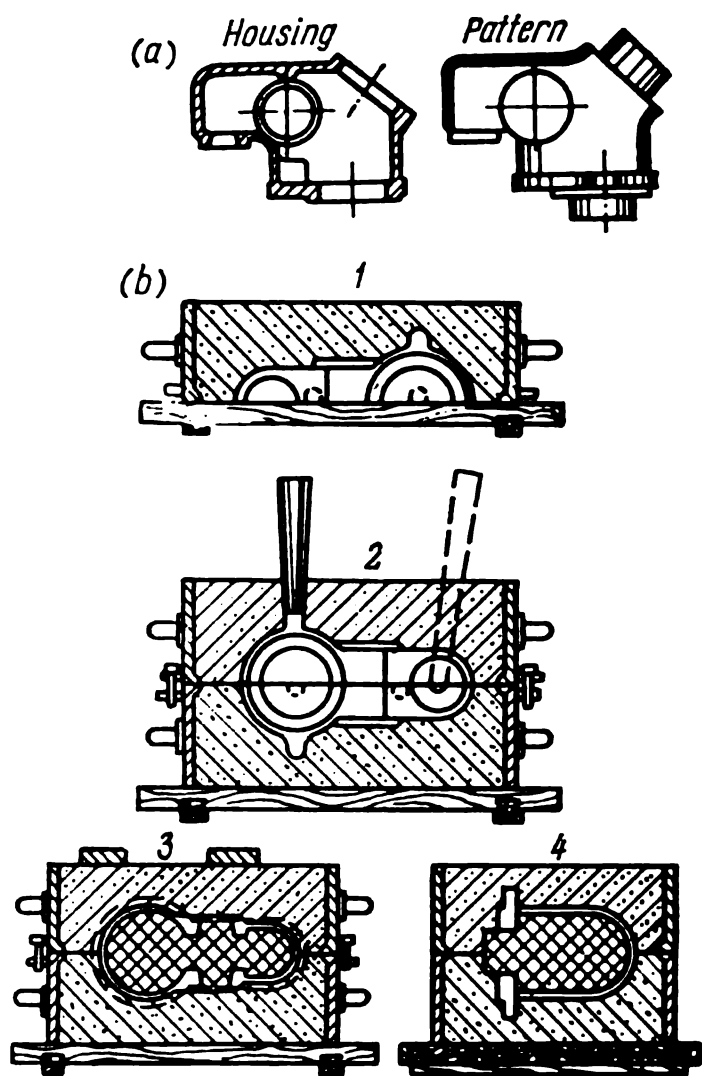


Fig. 65. Moulding an engine valve housing:  
(a) valve housing and its pattern; (b) making the mould; 1—drag flask; 2—cope and drag of the mould;  
3 and 4—sections of the assembled mould

Two-part moulding with a split pattern is employed to cast parts of medium complexity. It has much in common with the moulding process just described in which an unsplit pattern is used. The moulding of an engine valve housing is illustrated as an example in Fig. 65.

Half of the pattern and the drag flask are placed on a mould board. Further procedure is similar to that of the preceding example.

Intricate parts require moulds with two or more parting planes, otherwise it proves difficult to draw the component parts of the pattern. In such cases a flask comprising three or more parts is used to

make each mould. The middle part of a three-part flask is called the cheek.

*Stack moulding* is used to make small light castings; one advantage of this process is that it requires much less floor space in the foundry.

Two types of stack moulding are employed: upright (Fig. 66a) and stepped (Fig. 66b).

In upright stack moulding from 10 to 12 flask sections are arranged one above another, having a common sprue through which all the moulds are poured.

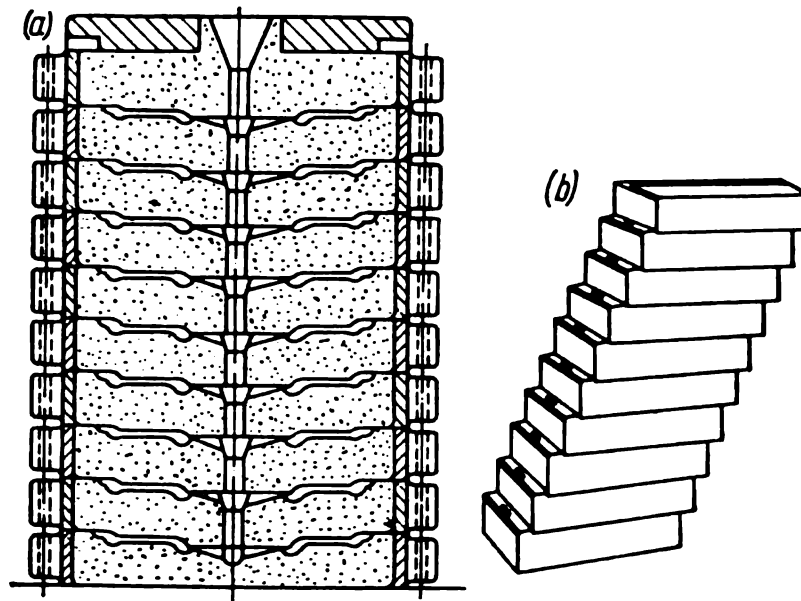


Fig. 66. Stack moulding:  
(a) upright; (b) stepped

In stepped stack moulding each flask section has its own sprue, the sections being arranged in steps. Each successive mould is offset from the one below it by the width of the pouring basin. Thus each mould is poured separately.

Match-plate patterns are used in snap flask moulding (Fig. 67). Match plates are designed so as to obtain an offset parting plane; this is done to avoid shifting of the cope and drag and to prevent molten metal from breaking out through the parting plane during pouring.

In this procedure the drag is placed on the metal match plate where it is filled with sand and rammed in the ordinary manner (Fig. 67a). For convenience the match plate may be put on the overturned cope 1. After ramming the drag 2, a bottom board 3 is placed on top and the whole mould is turned over to fill and ram the cope (Fig. 67b). Then the cope is lifted off (Fig. 67c) and, after it, the match-plate pattern 4 (Fig. 67d). Both halves of the mould are finished by hand and assembled (Fig. 67e). After this the cope and drag sections of the flask are

removed simultaneously from the mould (Fig. 67f). This presents no difficulty because of the tapered form of the flask. The mould is then delivered to the pouring zone where a steel jacket 5 is placed around it and a weight, in the form of a cast-iron plate with a hole for the sprue, is put on top.

Snap-flask moulding is extensively used for producing small castings on a large scale. Knock-out is much easier in this method and a significant economy is attained in the cost of flasks. Moulding sand consumption, however, is somewhat higher.

*Sweep moulding* on the foundry floor or in a pit using a form sweep template is resorted to when a part of large size is to be cast in a short time. The making of a pattern for flask moulding would be much more expensive and take a great deal of time. Sweep moulding excludes the use of expensive patterns and therefore costs about one half as much as moulding with patterns. Moreover, in moulding a heavy part in a flask, the pattern needed may be too cumbersome and difficult to remove from the mould.

Sweep moulding may be performed by two methods:

(1) with a turning sweep (template) rotating about either a vertical or horizontal axis to form surfaces of revolution (in moulding cylinders, bowls, etc.);

(2) with a drawing sweep pushed along a guide frame.

**Moulding machines.** The application of moulding machines enables labour productivity to be sharply increased, more accurate castings to be produced, costs to be reduced and a higher quality of product to be maintained.

Moulding machines pack the sand and draw the pattern from the mould. According to the method in which they compact the sand moulding machines are classified as squeezers, jolt machines and sand slingers.

*Squeeze moulding machines* are operated by compressed air at a pressure from 5 to 7 atm.

A schematic diagram of a top squeeze machine is shown in Fig. 68a. The pattern plate with pattern 2 is clamped on the work table 1 and flask 3 is placed on the plate. Then the sand frame 4 is placed on flask 3. The flask and frame are filled with moulding sand from a hopper located above the machine. Next the table lift mechanism is switched on and the flask together with the sand frame and pattern is lifted up against platen 5 of the stationary squeeze head 6. The platen enters the sand frame and compacts the moulding sand down to the upper edge of the flask (shown by a dash line). After the squeeze, the work table returns to its initial position. This machine moulding method is used for work that can be moulded in shallow flasks.

The principle of a bottom squeeze machine is shown in Fig. 68b. The pattern plate 2 with the pattern is clamped on work table 1.

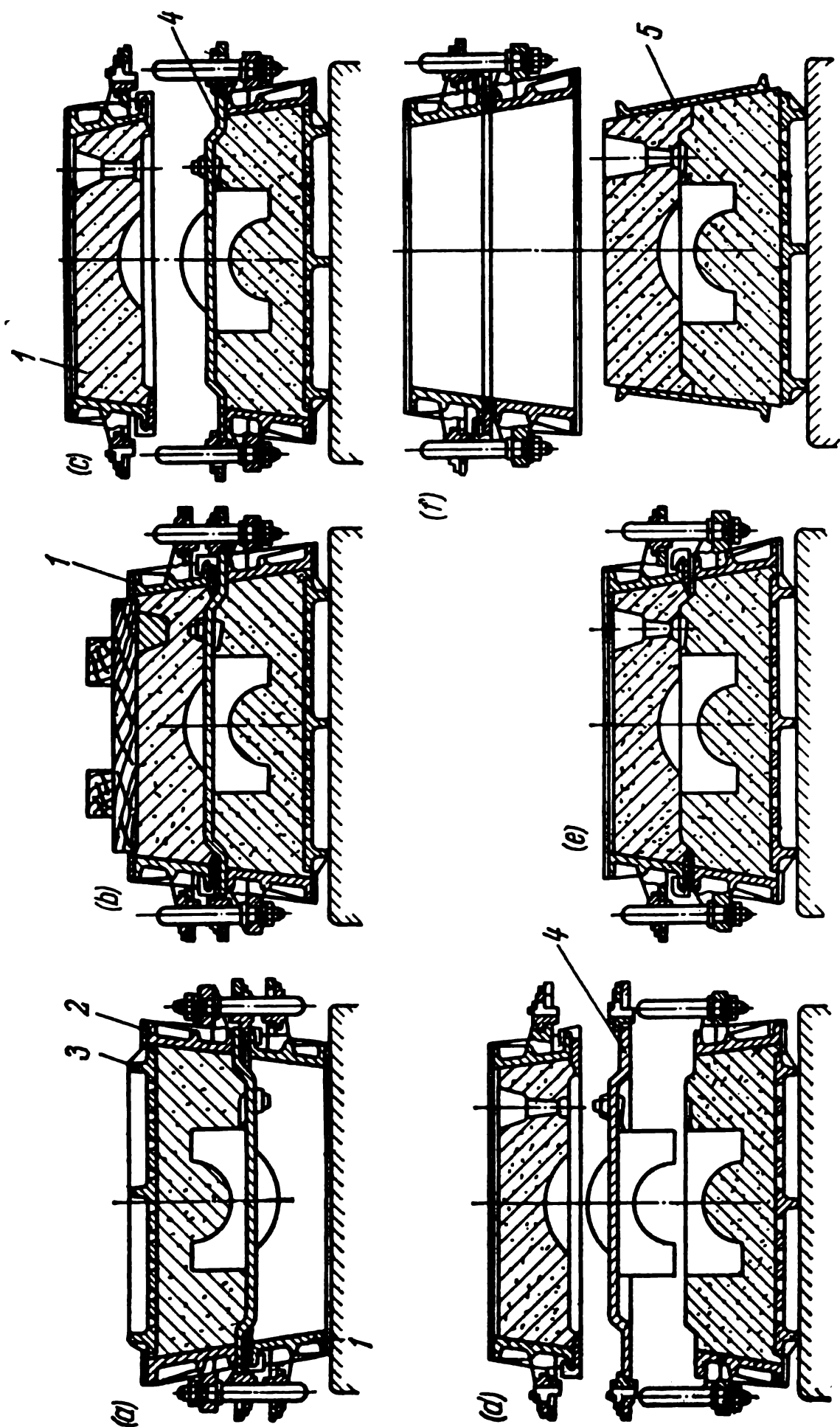


Fig. 67. Snap flask moulding

Flask 3 is placed on frame 7 of the machine and is filled with sand from a hopper. Next the squeeze head 6 is brought against the top of the flask and the lift mechanism is switched on. Table 1 with plate 2 and the pattern are pushed up to the lower edge of the flask (shown by the dash line). After this the table returns to the initial position.

Fig. 68c illustrates the principle of a plain shockless jolt moulding machine.

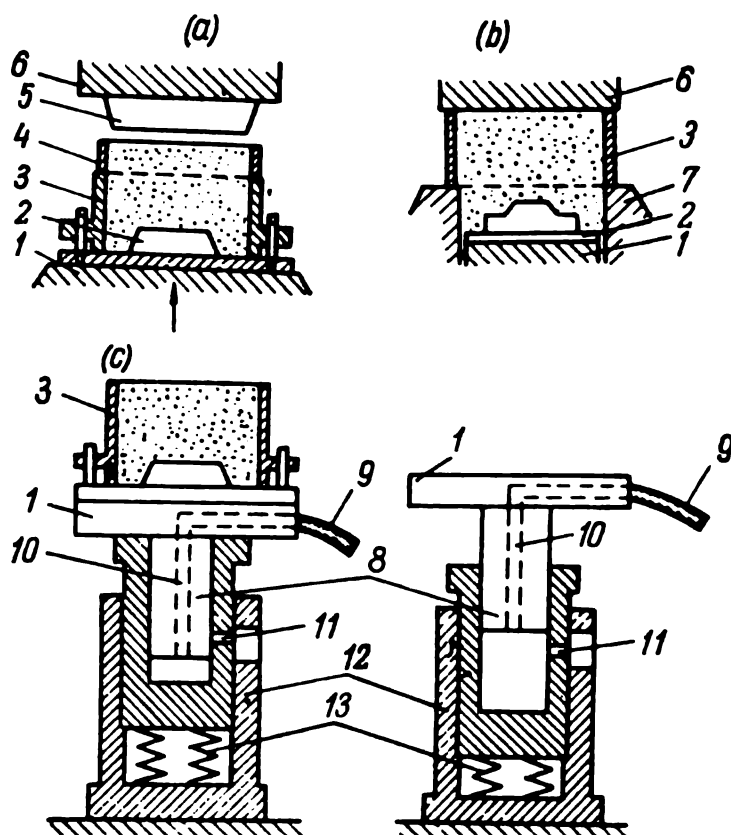


Fig. 68. Machine moulding principles

In the operation of a *jolt moulding machine*, table 1 with the pattern plate, pattern and flask 3, filled with moulding sand, is lifted by plunger 8 to a definite height when compressed air is admitted through hose 9 and channels 10. Next the table drops since the air is released through hole 11. In falling, the table strikes the stationary guiding cylinder 12 and this impact packs the moulding sand in the flask. Springs 13, by cushioning the table blows, reduce noise and prevent destruction of the mechanism and the foundation.

In machine moulding the pattern is drawn or stripped from the half mould by various methods. One method is shown in Fig. 69a where the rammed half mould 2 is raised by stripping pins 6 while the pattern 1 with pattern plate 3 remains on the table 5.

The drawing of pattern 1 through a stripping plate 4 is shown schematically in Fig. 69b. Pattern 1 with pattern plate 3 is lowered while the moulded flask and the stripping plate 4 remain stationary.

This method finds application in stripping high patterns. Another stripping plate procedure is shown in Fig. 69c, but here the finished half mould with stripping plate 4 are lifted by pins 6.

The pattern drawing principle incorporated in a roll-over moulding machine is illustrated in Fig. 69d. The moulded flask 2 together with pattern 1 and work table 5 is rotated 180° about their approximate centre of gravity and then pins 6 lift table 5 together with pattern 1 out of the mould.

The schematic diagram in Fig. 69e shows the operation of the rock-over pattern draw. The moulded flask 2 together with pattern 1 and table 5 is swung over by the arm onto the drawing table 7 which is then lowered with the mould away from the pattern.

**Sand slingers** (Fig. 70) fill flasks of any size and pack the moulding sand; they can be efficiently employed in both mass and piece production. These machines operate with a high output; one sand slinger can fill flasks, packing the sand, at a rate of 60 cu m per hour. Only these two operations are performed by a sand slinger; its disadvantage is that it does not draw the pattern or handle the mould in any way.

The principle of operation of the impeller head on the sand slinger is shown in Fig. 70a. The head consists of housing 1 in which blade 2 rotates rapidly. Moulding sand is fed by a belt conveyer to opening 3 in the end face of the housing where it is picked up by blade 2 and thrown in separate portions at a high speed through outlet 4 down into the flask under the head.

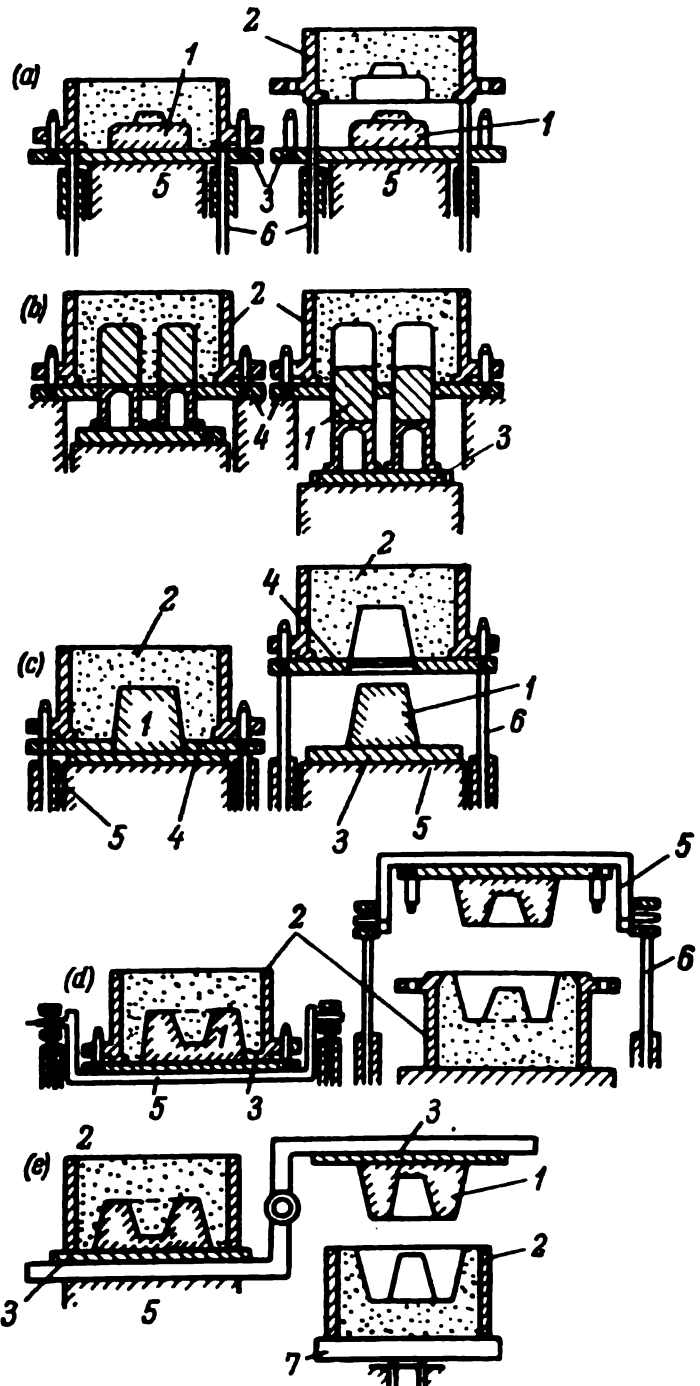


Fig. 69. Stripping the pattern in machine moulding



A motive-type sand slinger, used in both lot and piece production, is illustrated in Fig. 70b.

This sand slinger is mounted on truck 5 which travels along the shop on rails 6 and upper crane track 7. Conveyor 8, passing along the whole length of the bay, continuously supplies moulding sand to the intermediate hopper 9. From here the sand drops into the slinger hopper and onto belt conveyor 10 arranged on the upper swivel arm 11. The sand is transferred from upper conveyor 10 to lower belt conveyor 12 which carries it to slinger head 13.

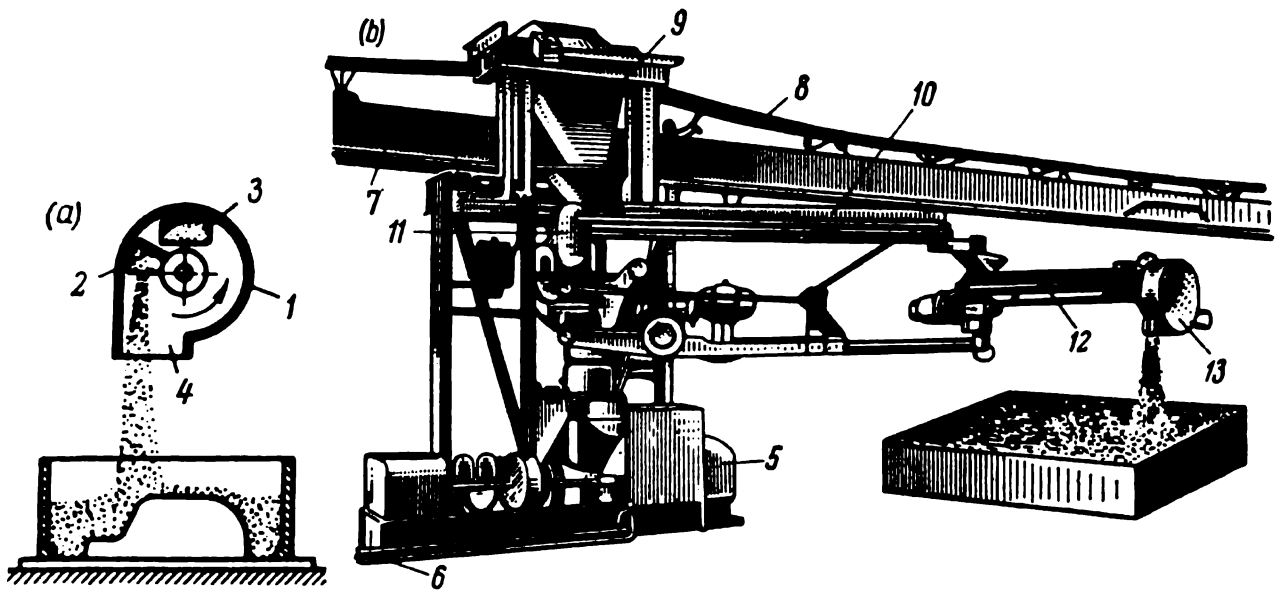


Fig. 70. Motive-type sand slinger:  
(a) principle of slinger head operation; (b) operation of the sand slinger

The slinger head is mounted on an arm that rotates about a horizontal axis thereby enabling the stream of sand to be directed to any part of the flask. Flasks to be filled are arranged along the line of travel of the slinger.

### 13-5. Coremaking

Through holes, recesses, projections and internal cavities are formed in a casting by setting sand cores into the mould; metal cores are less frequently used.

Cores are subject to severe conditions since after pouring the mould they are surrounded on all sides (except for the ends) by molten metal. Consequently, they must possess exceptionally high strength, good permeability, ample refractoriness and sufficient collapsibility. A lack of this latter ability to decrease in volume may lead to the formation of cracks in the casting as the metal cools.

According to their *size and strength requirements* cores may be classified as:

- (1) medium and heavy clay-and-sand cores made of silica sand, clay and organic admixtures;
- (2) light and sometimes heavy sand cores made of silica sand and organic binders;
- (3) especially heavy high-strength cores made of clay and brick;
- (4) metal cores of simple shapes.

Cores are made manually or with machines. Cores are made by hand in core boxes or by using sweeps; only core box methods are employed for machine coremaking.

Coremaking consists of the following sequence of operations: moulding a green core, baking, finishing and coating. If a core is made of two or several pieces they are assembled together after baking by pasting or other methods.

Some cores require internal reinforcing within the sand in the form of steel wire or grey-iron castings to increase their strength.

Reinforcement (core irons) is most often made of annealed low-carbon steel wire up to 8 mm in diameter. Heavy cores with a large cross section may be reinforced with cast-iron grids.

Cores for pipes and other parts which are solids of revolution may be made by turning clay to a template. Such cores are reinforced by a barrel in the form of a steel pipe with holes along its length for gas escape.

It is important to properly locate the core iron in the core. It should never be too close to the surface, otherwise a thin layer of sand may break off.

Core venting, i. e., the provision of vent holes to improve the permeability of the core, requires special attention. Vent holes may be made by piercing the sand with a wire from the core print end so that the holes do not reach the external surfaces of the core as otherwise molten metal may get into them.

Additional vent holes are made by moulding rods in the sand which are withdrawn before the core is stripped leaving straight holes.

The process of making a simple core in a split core box is shown in Fig. 71.

Cores of intricate shape are made of separate pieces pasted together after baking.

Round cores over 500 mm long and of a diameter up to 60 mm and more can be made by using sweeps or templates;

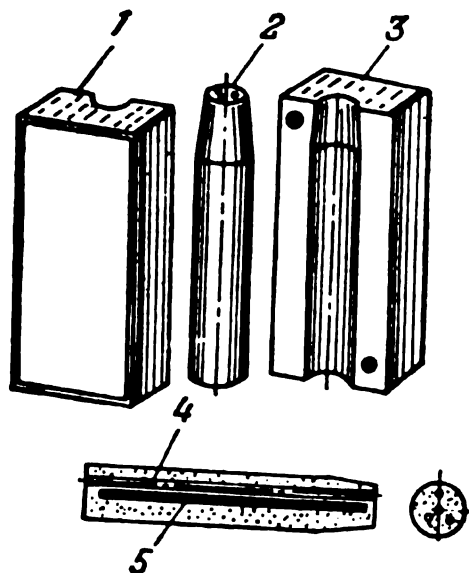


Fig. 71. Core box and vent hole arrangement in a core:

1 and 3—halves of the core box;  
2—core; 4—vent holes; 5—core iron (2-mm wire)

cores which are not solids of revolution can be made with a drawing sweep.

The operations of coremaking are also performed by various machines. According to the methods employed to compact the core sand, coremaking machines are classified as die extrusion (stock core), squeeze and jolt machines, sand slingers or throwers, core blowers and special machines.

Heavy cores are handled in the foundry and set into the mould by means of a crane.

### 13-6. Gating Systems

**Purpose and types of gating systems.** A *gating system* is a system of channels, including the pouring basin, by means of which molten metal is delivered to the mould cavity.

The functions of a gating system are:

(1) to provide continuous, uniform feed of molten metal, without turbulence, to the mould cavity;

(2) to supply the casting with liquid metal during solidification and shrinkage;

(3) to prevent slag, sand and other nonmetallic inclusions from entering the mould;

(4) to prevent destruction of the mould walls by the stream of metal.

A *standard gating system* for cast iron (Fig. 72) consists of the pouring basin (or sprue cup) 1, downsprue 2, runner 3 and ingates 4. Flow-off 5 is provided whenever required for grey-iron castings, or a riser is made in other cases.

The two main types of gating systems differ in the manner in which the metal is delivered to the mould cavity. They are:

(1) *Horizontal gating* where the metal is conducted into the mould cavity in a horizontal plane (Fig. 73), usually in the parting plane of the flask.

(2) *Vertical gating* in which the metal enters the mould cavity along a vertical plane (Fig. 74).

**Design of gating systems.** The dimensions of a gating system are determined on the basis of the cross-sectional area of the ingates. The dimensions of all the other elements (passages) of the system depend upon the size of the ingates.

The rate at which the mould cavity is filled with metal depends upon the ingate cross section.

Factors to be considered in assigning the ingate cross section are the wall thickness, configuration, overall size and weight of the casting. Larger ingates are required for heavier castings than for light ones; pouring will be prolonged if the metal passes through narrow passages, and the metal may freeze before it fills the mould.

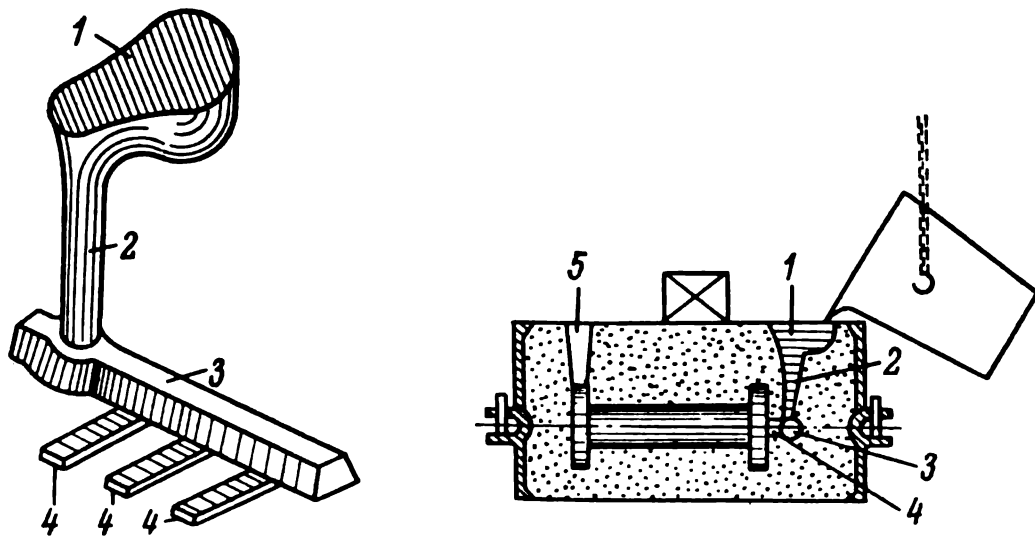


Fig. 72. Standard gating system for cast iron

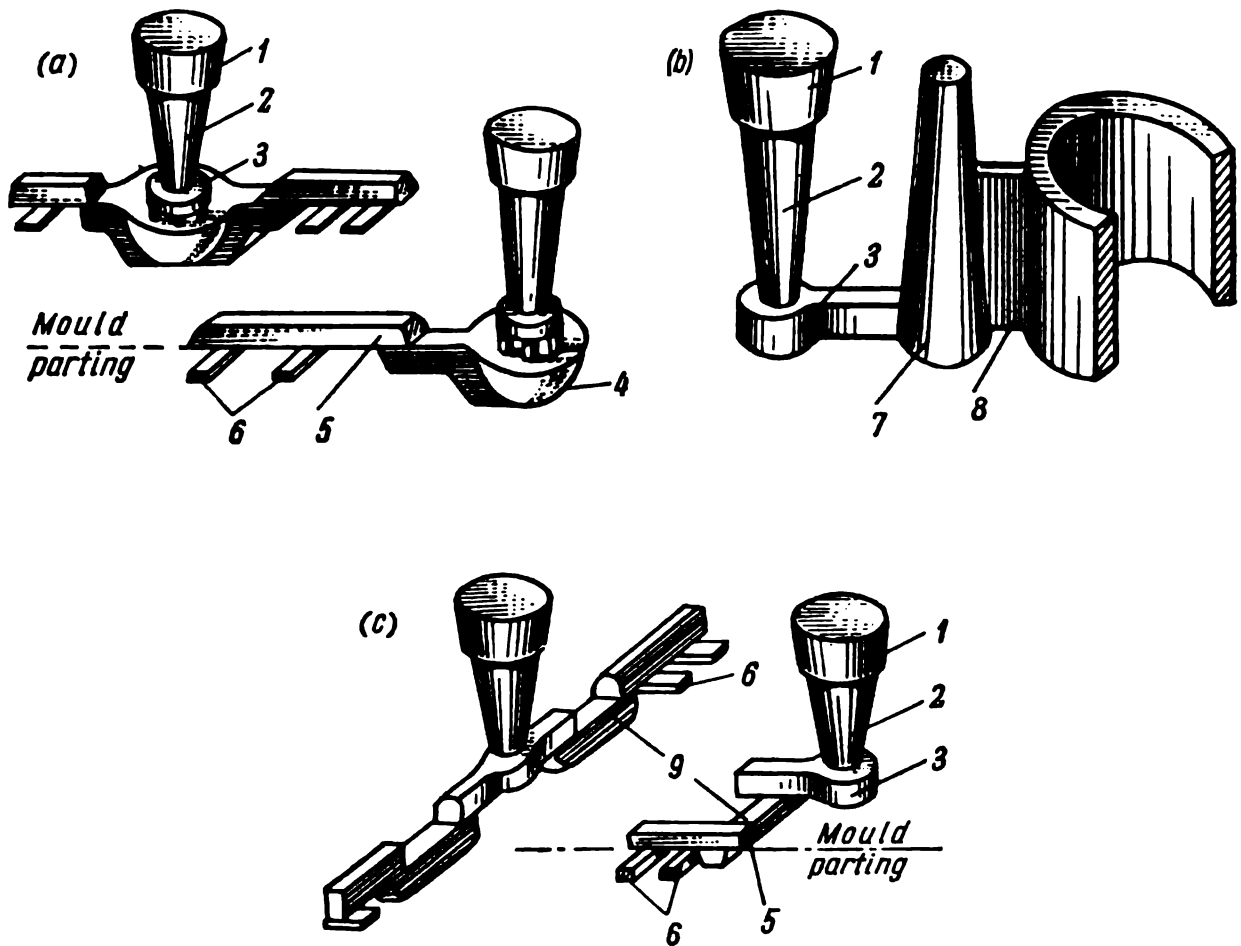


Fig. 73. Horizontal gating systems:

(a) with a strainer; (b) with a knife gate; (c) retarding type with overlapped runners; 1—sprue cup; 2—downsprue; 3—sprue base; 4—strainer gate bottom; 5—runner; 6—ingates; 7—vertical runner; 8—vertical knife gate; 9—overlapped runners

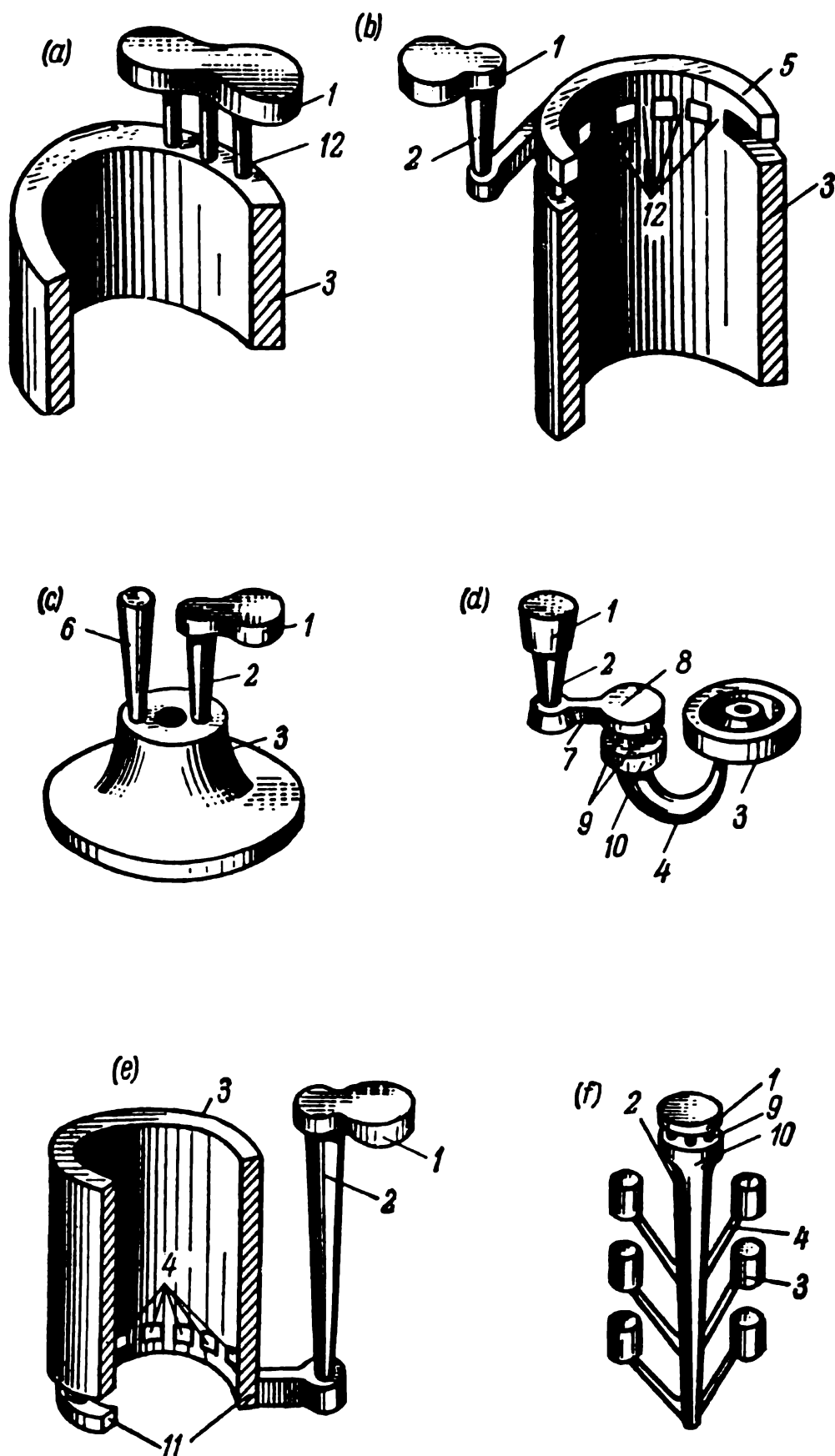


Fig. 74. Vertical gating systems:

(a) shower gate; (b) shower gate with runner; (c) simple top gate; (d) horn gate with strainer; (e) bottom gate with ring runner; (f) bottom gate for light castings; 1—pouring basin or sprue cup; 2—downsprue; 3—casting; 4—horn gate; 5—ring gate; 6—flow-off; 7—runner; 8—strainer gate top; 9—strainer gate; 10—strainer gate bottom; 11—ring runner; 12—pencil gate

Excessively long passages should also be avoided since during pouring the metal may thicken and lose its fluidity. This may lead to rejects due to misruns, cold laps and coldshuts.

The pouring rate depends, not only upon the passage cross section, but also upon the pressure of the metal in the gating system; the higher the downsprue, the more rapidly the mould is filled with metal.

The following formula, applied to compute the cross section of the ingates, is based upon the conditions of fluid flow in closed channels (flow in pipe):

$$F = \frac{g}{0.31\mu t \sqrt{H_e}} \text{ cm}^2, \quad (64)$$

where  $F$  = total ingate cross-sectional area, sq cm,

$g$  = weight of the casting, kg,

$\mu$  = pouring factor (ratio of casting weight to metal poured) which ranges from 0.27 to 0.55 for grey-iron castings, 0.3 to 0.41 for steel castings and 0.6 to 0.7 for nonferrous castings,

$t$  = pouring time, sec,

$H_e$  = effective head of molten metal, cm.

The effective head  $H_e$  is determined by the formula

$$H_e = H_{max} - \frac{H_g^2}{2H_c} \text{ cm}, \quad (65)$$

where  $H_{max}$  = maximum head (level of liquid metal above ingate level),

$H_g$  = height of the casting above the ingate level,

$H_c$  = total height of the casting.

The cross sections of the downsprue and runner depend upon that of the ingates and are determined from the ratio 1.4 : 1.2 : 1 (ratio of downsprue to runner to total ingate cross-sectional area).

## Chapter 14

### FOUNDRY FURNACES, TYPES AND PROPERTIES OF CASTINGS, POURING AND CLEANING CASTINGS

#### 14-1. Charging Materials

*In the production of grey iron and steel castings the charge consists of metallic materials, fuel and fluxes.*

The metallic part of the charge is made up of definite quantities of pig iron of various grades, cast-iron and steel scrap, foundry scrap

(gating, splashes and spills, rejects and chips) and a small amount of ferroalloys. Foundry coke and, sometimes, anthracite are used as fuel for melting grey iron in cupolas. Fluxes used in melting both grey iron and steel may be limestone, dolomite, fluorspar, apatite ore and open-hearth slags. The function of the fluxes is to lower the melting point of the slag formed in melting.

*In the production of nonferrous castings* the metallic part of the charge comprises the corresponding primary and secondary metals and alloys. Primary metals are produced in metallurgical plants. Secondary metals and alloys are those obtained by remelting and refining nonferrous scrap and chips in furnaces and casting the molten metal into flat ingot moulds.

A so-called master alloy, a specially prepared alloy of two or more metals, is frequently added to the metal charge for making nonferrous castings.

Fluxes used in melting nonferrous metals are usually chlorous and fluoric salts of alkali and alkaline-earth metals, e. g., barium chloride is used in melting magnesium- and aluminium-base alloys.

## 14-2. Foundry Furnaces

The size of foundry furnaces and their output must suit the requirements in molten metal of the given foundry.

Various types of foundry furnaces are available to melt different kinds of metals. Converters, open-hearth and electric furnaces for melting steel were described in Part One of this book. Only certain furnaces used in foundries to obtain molten grey iron, some of the nonferrous metals and their alloys will be discussed in the following.

*Structural features of a cupola.* The cupola is the most widely used furnace in the foundry for melting grey iron.

A cupola is a shaft furnace of cylindrical shape (Fig. 75) erected on legs or columns 1. The cupola shell 2 is made of steel plate 8 or 10 mm thick. The interior is lined with refractory fireclay brick 3. Cast-iron plates 4 encircling the throat protect the lining in the upper part of the cupola stack against damage caused by lumps of the charge.

The solid materials, coke 5 and metal 6, are charged into the cupola from the charging floor through charging hole 7. Charging is mechanised in large foundries where drop-bottom buckets 8 are employed.

Air for combustion of the fuel is delivered from blower 19 through tuyeres 9 arranged above the cupola well. Modern cupolas often have three rows of tuyeres arranged in checkerboard fashion.

The products of combustion (flue gases) pass out through stack 10 and spark arrester 11 (dry or water-sprinkled type).

The lower part of the cupola, from sand bottom 12 to tuyeres 9, is called the well 13.

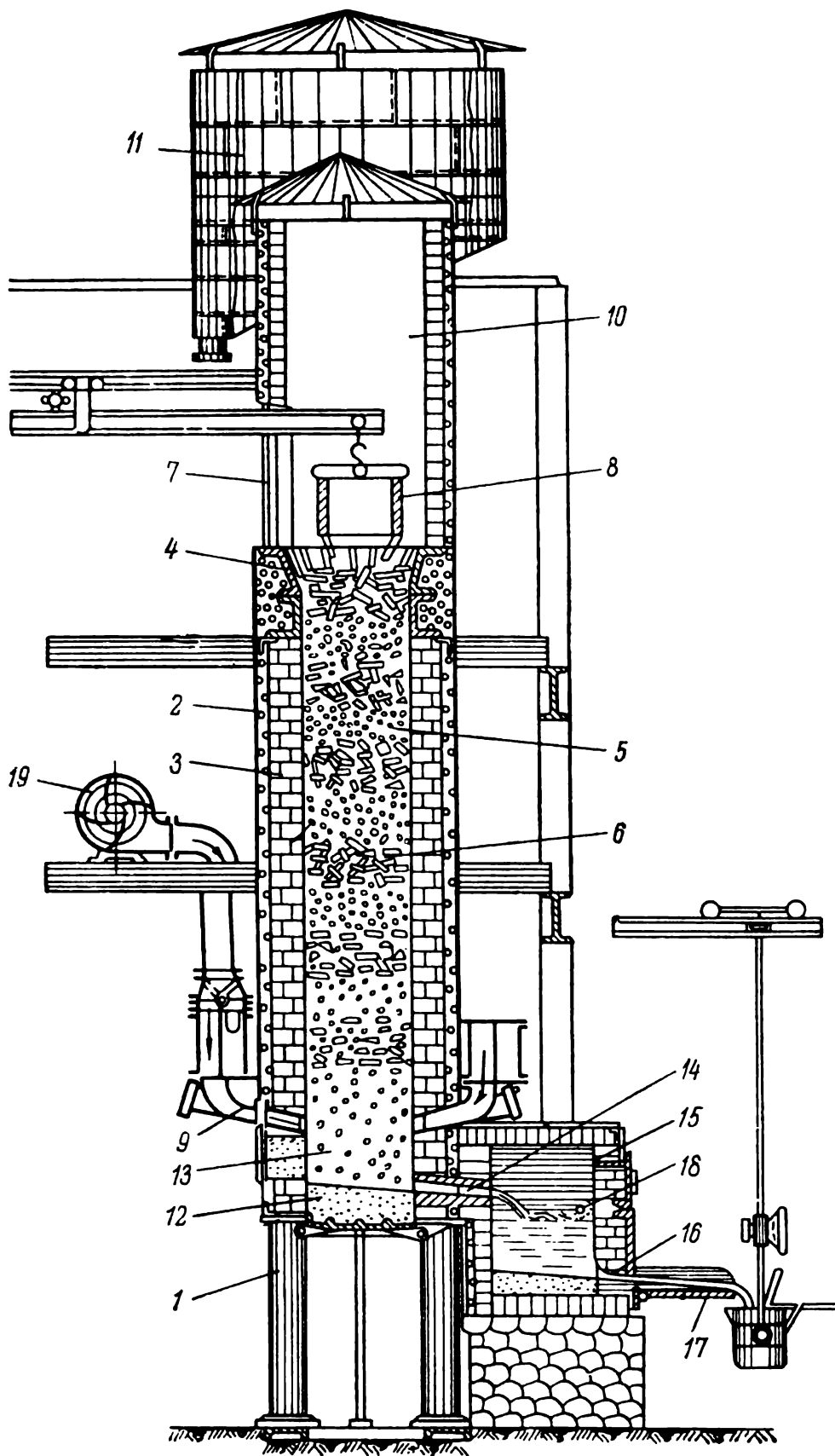


Fig. 75. Cross section of a cupola



The bottom is rammed of weak moulding sand possessing high refractoriness. After each heat the bottom is dropped. A taphole 14 is provided at the lower part of the sand bottom to let the molten iron into the receiver 15. The diameter of the receiver in the clear is equal to or slightly larger than that of the cupola stack. From the receiver the metal is let out through taphole 16 along tapping spout 17 into the ladle. Slagging is done, as required, through cinder notch 18.

The principal dimensions of a cupola are selected on the basis of empirical data. Thus, the cross-sectional area  $F_c$  of a cupola depends upon the designed hourly output and is determined from the formula

$$F_c = \frac{\pi d^2}{4} = \frac{Q}{Q_1} \text{ m}^2, \quad (66)$$

where  $d$  = cupola diameter in the clear, m,

$Q$  = designed cupola output, tons per hour,

$Q_1$  = specific output per sq m of shaft cross-sectional area, tons per hour. As a rule  $Q_1 = 6$  to 8 tons per hour.

The useful height of a cupola (distance from the axis of the main tuyeres to the lower edge of the charging hole) depends upon the diameter and is designed according to the ratio  $H : d =$  from 3 to 5.

The cupola height directly affects the melting rate, fuel consumption and the temperature and quality of the molten metal. If the cupola is too high the coke may be crushed as the charge drops. If it is too low the metal is not heated to a sufficient degree, the draught is reduced and the cupola output is decreased.

The cupola tuyeres should deliver a uniform blast sufficient for complete combustion of the fuel. This requires that their total cross section should be in a definite relation to the cupola cross section. Cupola output and the temperature and quality of the metal depend upon the tuyere design. The total cross-sectional area of the tuyeres should equal from  $\frac{1}{6}$  to  $\frac{1}{8}$  of the area of the cupola in the clear. Usually tuyeres have a size of  $50 \times 150$  or  $100 \times 300$  mm.

The tuyeres of receiver cupolas should be as near as possible to the sand bottom. In cupolas without receivers the position of the tuyeres above the bottom is determined by the amount of molten metal that it is necessary to accumulate in the well before tapping.

The height of a well for a receiver cupola should not exceed 150-200 mm; it should not be over 450 mm for cupolas without receivers.

**Cupola operation.** A newly built cupola (or one after overhauling or current repairs) should be thoroughly dried before firing.

In firing a cupola, i. e., in starting a heat, a fire of kindling wood is ignited on the sand bottom. When the wood is burning well coke is dumped into the well from above in several portions making sure that the coke begins to burn too. Then coke is added to a level slightly above the tuyeres and the air blast is turned on at a lower than

normal blowing rate. This intensifies coke combustion. Then new portions of coke are charged into the cupola to reach a height 700 to 800 mm above the upper row of tuyeres. This layer of coke resting on the sand bottom before beginning the heat is called the *bed charge*.

The height of the bed charge is very important to the cupola process; it affects the temperature, melting rate and chemical composition of the grey iron tapped from the cupola. Coke burns within this bed and produces a zone where the highest temperatures are developed (melting zone of the cupola). Metal and flux are charged on the coke bed.

As a rule the height of the bed charge is based upon previous practice and depends upon the size and output of the cupola.

As soon as the coke bed is thoroughly ignited, pig iron and scrap may be charged. The weight of the metal charge should be from 10 to 15 per cent of the hourly output of the cupola.

A certain amount of limestone, the flux used to form the cupola slag, is added above the first metal charge. Upon being heated in the cupola the limestone is decomposed into lime ( $\text{CaO}$ ) and carbon dioxide ( $\text{CO}_2$ ). The lime neutralises (binds) the silica introduced into the cupola with the coke ash and sand or formed in the oxidation of silicon; the  $\text{CO}_2$  is carried out by the flue gases.

The reaction between the silica and lime reduces the melting point of the slag and increases its fluidity.

The added limestone should constitute about 2 or 3 per cent of the metal charge by weight.

After the first layers of pig iron and flux, the first working charge of coke is made; it should weigh from 8 to 12 per cent of the metal charge. As the coke in the bed charge burns, that of the working charge gradually drops down together with the molten materials under it to replace the burned coke of the bed charge. This makes it possible to maintain a constant rate of coke combustion in the bed charge.

Alternate charges of coke and iron are made up to the charging hole level.

Full blast, delivered to the tuyeres from the blower through the blast inlet pipe, is turned on after completely charging the cupola.

The required blast pressure is determined from the empirical formula

$$P = 64 \sqrt{q_0 F_c} \text{ mm of water column.} \quad (67)$$

where  $q_0$  = specific volume of the blast per minute per sq m of cupola cross section (100 to 130 cu m),

$F_c$  = cross-sectional area of the cupola in the clear, sq m.

The taphole and cinder notch are left open at first to facilitate coke combustion in the well. When the first drops of molten iron

appear, the taphole is plugged with a wad of clay after which the coke burns only at the tuyere level and slightly above. In this zone there is a surplus of oxygen from the blast and the main reaction is the combustion of carbon:  $C + O_2 = CO_2 + 94,220$  cal per mol.\* Heat evolved in this reaction is expended in heating up the coke, gases and the charge in the cupola.

Highest temperatures are developed in the zone of complete combustion of the coke; they reach  $1600^\circ$  to  $1700^\circ\text{C}$ . Above this zone, in the so-called *melting zone*, the temperature falls slightly but is ample to melt the metallic part of the charge and to form slag.

The drops and trickles of molten iron are overheated as they flow down and reach the zone of complete combustion of the coke where there is surplus air and a large quantity of  $CO_2$ . Therefore, a certain liquid metal loss through burning is observed near the tuyeres, as in the Bessemer process.

There is practically no surplus oxygen below the tuyere zone since the blast does not reach this part of the cupola. Therefore, no further oxidation of the metal is observed in the well where the molten iron accumulates. On the other hand, in the well the metal is saturated with sulphur and, to some extent, carbon from the coke. The less the height of the well, the less time the molten iron is in contact with the coke and the less carbon and sulphur it picks up.

As a result of the above-mentioned chemical processes, the composition of pig iron melted in the cupola undergoes the following changes: the silicon content is reduced by about 15 per cent, manganese by up to 20 per cent, the carbon content remains practically constant and the sulphur content is increased by from 30 to 60 per cent, depending upon its content in the coke and upon the coke consumption.

The iron oxidation products ( $SiO_2$ ,  $MnO$ ,  $FeO$ , etc.) pass into the cupola slag.

Gases evolved in the tuyere zone (a mixture of  $CO_2$ ,  $CO$ ,  $N_2$ ,  $H_2O_{vap}$ ,  $SO_2$ , etc.) are conducted upwards to the spark arrester, thus passing through the whole charge. They give up part of their heat to the charging materials (pig iron, fluxes and coke) and their composition is changed to some extent. Usually, the temperature of the outgoing gases ranges from  $300^\circ$  to  $400^\circ\text{C}$ ; this indicates that heat is well utilised in the cupola.

As the coke burns and the pig iron melts, space is freed at the lower part of the cupola and the stack contents gradually settle. Consequently, new charges must be added in layers through the charging hole in the above-mentioned order to keep the cupola working continuously.

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\* The reaction:  $CO_2 + C = 2CO - 41,080$  cal per mol develops to a certain extent in the upper layer of the coke bed and partly in the overlying first working charge, thereby reducing the charge temperature to some degree.

The liquid products formed in cupola operation — iron and slag— are accumulated in the well in the form of separate immiscible layers which are let out through the taphole and slag notch.

The output of a cupola is defined as the tons of liquid metal obtained per hour of the heat; it depends not only upon the cross-sectional area of the cupola, but upon the intensity of coke combustion as well. By the intensity of coke combustion is meant the tons of coke burned per square metre of the cross-sectional area of the cupola in unit time.

Measures that increase cupola output and reduce fuel consumption are: (1) oxygen enrichment of the air blast (up to 30-35% O<sub>2</sub>) and (2) utilisation of the heat of the outgoing gases to preheat the blast.

The great advantage of the cupola over other foundry furnaces is in its continuous operation.

Cast irons of various chemical compositions can be obtained in a cupola.

**Flame furnaces.** Pig iron and nonferrous metals can be melted in reverberatory and rotating flame furnaces.

The air, or reverberatory, furnace shown in Fig. 76a is used to melt bronze, brass and malleable iron. It is called reverberatory because the furnace roof reflects the flame into the metal which is melted and overheated. Such furnaces are fired with coal, pulverised solid fuels, masout or gas.

Rotating-drum furnaces of the Mechta type (Fig. 76b) are frequently employed to melt nonferrous metals. These furnaces have two chambers. Masout, delivered through a burner at a pressure of 150 to 200 mm of water column, is burned in the first chamber. Flue gases pass through a channel into the second chamber where the metal is melted. Combustion products pass out of the furnace through a flue duct.

Rotating-drum furnaces with a capacity up to 300 kg are revolved manually with a handwheel; furnaces of larger capacity—by special mechanisms.

In a 300-kg furnace bronze is melted in one hour; pig iron—in 1.5 to 2 hours.

**Crucible and pot furnaces.** Chamotte or graphite crucibles are used for melting copper-base alloys. Aluminium- and zinc-base alloys are melted in steel or cast-iron pots.

Crucible and pot furnaces may be fired by either liquid, gaseous or pulverised fuels.

The products of fuel combustion heat the walls of the crucible or pot which transmit the heat to the metal inside. Thus, in most cases, the products of combustion do not come into direct contact with the metal, the metal does not become saturated with flue gases and high-quality castings are produced.

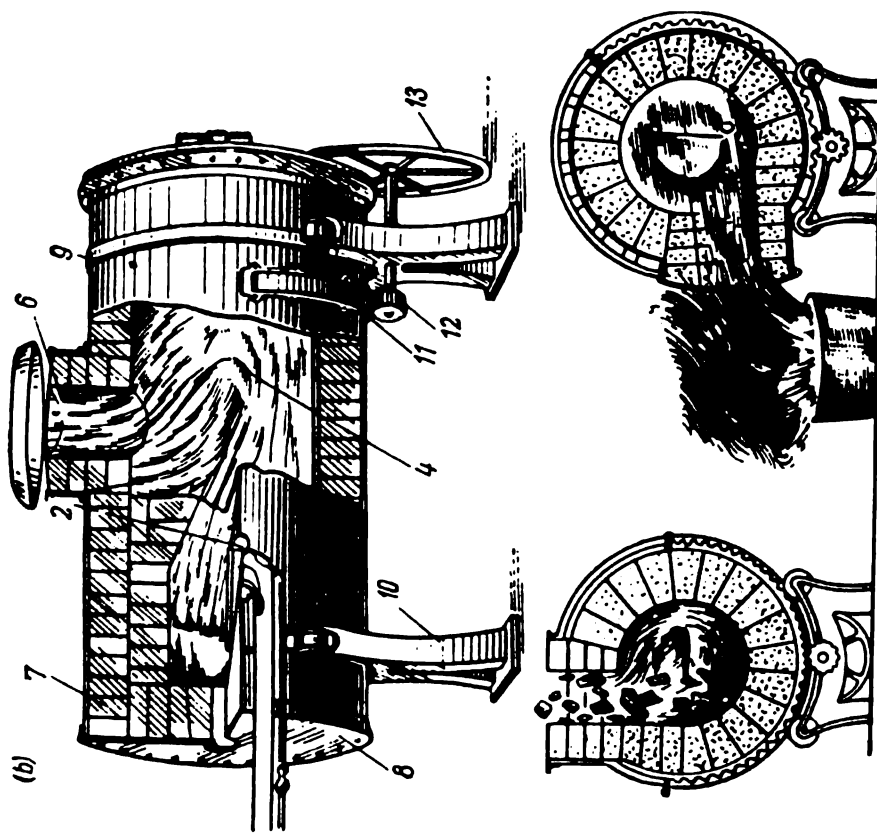


Fig. 76. Melting furnaces:

(a) air, or reverberatory, furnace; (b) rotating-drum furnace; 1—air inlet; 2—burners; 3—combustion chamber; 4—melting chamber; 5—charging hole; 6—flue; 7—shell lining; 8—end wall; 9—steel bands; 10—cast-iron supports; 11—segment gear of the tilting device; 12—pinion; 13—handwheel

Crucible and pot capacities range from 30 to 150 kg; the melting time depends on the metal and the type of fuel used.

Tilting-pot furnaces, employed at the present time, are mounted above floor level on supports. The furnace is tilted on trunnions to pour the molten metal.

A disadvantage of crucible and pot furnaces is their high fuel consumption.

In steel foundries the steel is melted in open-hearth furnaces, electric-arc furnaces of suitable tonnage and in side-blown converters.

**Electric furnaces.** The molten metal does not come into contact with the products of combustion in electric furnaces, and thus the loss of the constituent elements is reduced and metal of high quality is obtained for founding purposes.

The following types of electric furnaces are used in foundry practice:

- (1) electric-arc,
- (2) resistance,
- (3) induction.

Most widely employed of these are *electric-arc furnaces* in which the heat radiated by an arc is used to melt the metal.

Electric-arc furnaces can be used to melt, not only steel, but grey iron as well and certain nonferrous metals and alloys.

The electric-arc furnaces used to make steel and remelt pig iron in foundries are usually built with a capacity up to 10 tons. They operate on three-phase current and consume from 600 to 850 kWh of electric energy to produce one ton of steel from a solid charge. The time required for one heat, or melt, varies with the furnace capacity and may reach 4 hours.

*Rotary-drum indirect-arc furnaces* are commonly used for melting copper-base alloys. A furnace of this type is illustrated in Fig. 77a. The liquid metal is inside a horizontal, refractory lined drum 1 which rotates about its axis on rolls 2. Two graphite or carbon electrodes 3 between which the arc is struck are inserted in the centres of the end walls. Materials are charged and the metal is poured through the charging door 4.

Aluminium-base alloys are melted in *electric resistance furnaces*. A schematic diagram of such a furnace is shown in Fig. 77b. The metal is charged into the melting chambers 5 with inclined hearths arranged at both ends of the furnace. Resistance coils 6 are heated by an electric current and transmit the evolved heat to the metal in the melting chambers. The metal melts and flows down into receiver 7. When required, the furnace is tilted by rotation on rolls 2 and the molten metal is poured into a ladle through spout 8.

*Induction furnaces* possess many advantageous features from the point of view of foundry practice. Metal can be heated to very high temperatures in these furnaces without the danger of localised over-

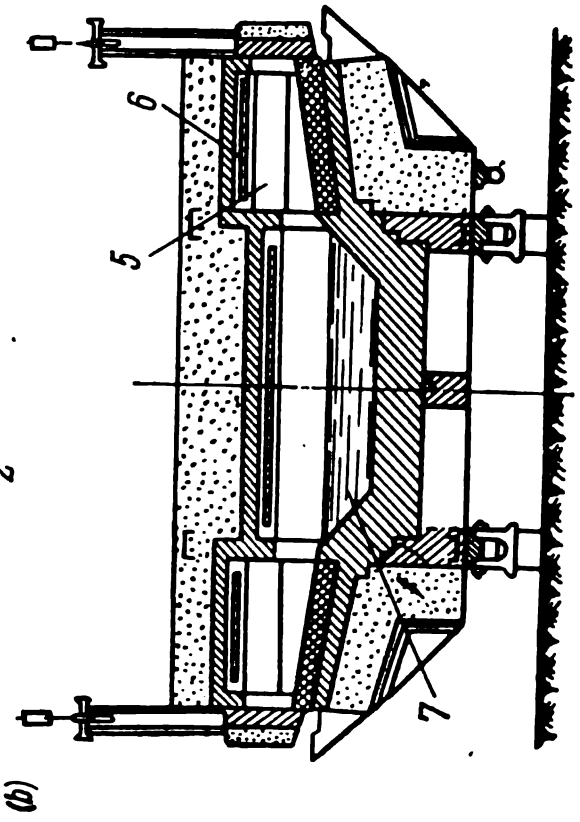
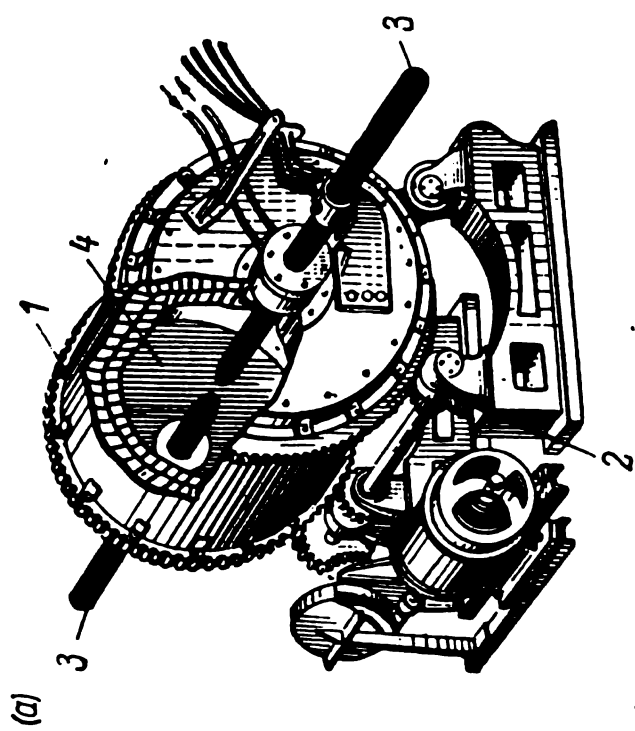
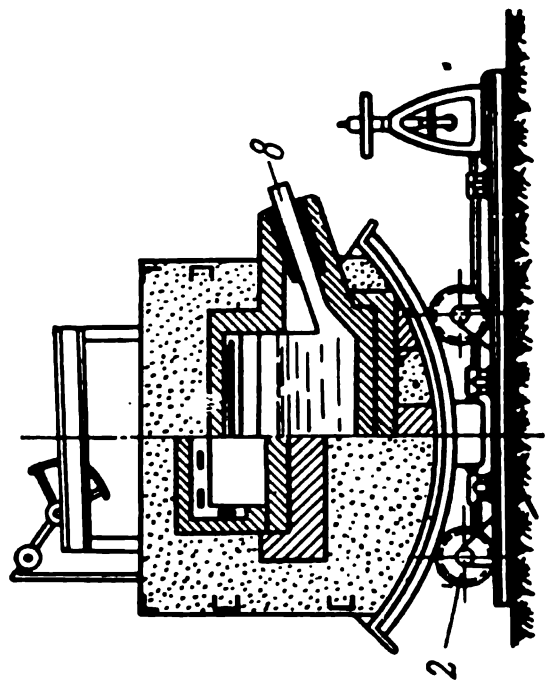


Fig. 77. Electric furnaces for melting nonferrous metals:  
 (a) rotary-drum indirect-arc furnace; (b) resistance furnace



heating that is sometimes observed in electric-arc furnaces. The composition of the gas atmosphere in induction furnaces may be controlled or a vacuum may be set up if necessary. The motor effect of the electromagnetic forces in the bath of an induction furnace produces a circulatory motion, or automatic stirring and mixing, which facilitates better interaction between the metal and slag, and metal of more uniform composition is obtained. Mixing the metal in the bath also has a favourable effect on its degasification and upon the removal of nonmetallic inclusions. Induction furnaces have simple controls; they can be readily started up or stopped.

These furnaces are employed to melt steel, iron, bronze, brass and aluminium-base alloys. They are especially widely used to melt precision alloys, i. e., alloys of exact composition based on iron and other metals.

The capacity of induction furnaces ranges from several kilograms to 4 or 6 tons.

### 14-3. Types and Properties of Iron and Steel Castings

**Grey iron.** Parts of various shapes and weights are cast of grey iron. The extensive application of grey cast iron in the engineering industries is based on its expedient combination of good castability with fair mechanical properties.

The mechanical properties of grey iron castings can be improved by inoculating the liquid metal. Inoculation is done by adding ferrosilicon, calcium silicide or other inoculants to the liquid metal in the cupola spout or ladle in amounts from 0.1 to 0.5 per cent of the iron by weight. These additions deoxidise the iron and hinder the growth of the precipitated graphite.

Grey iron castings are identified in the U.S.S.R. by a letter and number system based on their mechanical strength; for example, in the designation CЧ18-36 the Russian letters CЧ represent the first letters of the Russian words "серый чугун", meaning grey cast iron. The first number indicates the average value of the tensile strength and the second, the bending strength, both expressed in kg per sq mm.

There are several methods for improving the quality of the iron in castings.

If, for example, 30 to 40 per cent steel scrap is added to the cupola charge, the total carbon content is reduced to 2.8-3 per cent and *pearlitic* (semi-steel) high-strength cast iron is obtained. When the carbon content of pearlitic cast iron is reduced it is necessary to increase its silicon content to promote graphitisation.

Pearlitic iron castings find applications in the manufacture of critical parts in engineering due to their high strength (their tensile strength is increased to 40-45 kg per sq mm).



Another type of high-strength cast iron can be produced in foundry practice by making ladle additions of magnesium. High-strength spheroidal graphite cast iron (also called ductile cast iron) is identified by the Russian letters ВЧ followed by two numbers. The first indicates the tensile strength of a cast test specimen, in kg per sq mm, and the second—its relative elongation in per cent. The most widely used grades of Soviet high-strength cast iron are ВЧ50-1.5 and ВЧ45-5.

Good machinability is observed in high-strength cast irons.

**Malleable cast iron.** Malleable iron is produced in the foundry by annealing (malleablising) white cast iron for 50 to 100 hours at a temperature from 900° to 1050° C.

According to the annealing procedure applied, either of two types of malleable iron is obtained—pearlitic (white heart) or ferritic (black heart).

Pearlitic malleable iron is produced by annealing white iron in an oxidising atmosphere. This is achieved by packing the white iron castings in metal boxes, imbedded in iron ore, and putting the boxes in a flame or electric resistance furnace. At the high temperature the iron carbide (cementite) in the cast iron decomposes according to the reaction  $\text{Fe}_3\text{C} \rightarrow 3\text{Fe} + \text{C}_{\text{temper}}$ . At the same time the surface of the castings is decarburised to some extent by the action of the oxygen in the ore.

After malleablisation castings of pearlitic cast iron have a bright silvery fracture.

Ferritic (black heart) malleable iron is most widely used in the U.S.S.R. It is produced by annealing white iron castings in a neutral medium without adding ore to the boxes. Only cementite is decomposed in this procedure;

the metal is not decarburised. Ferritic malleable cast iron has a velvety-black fracture.

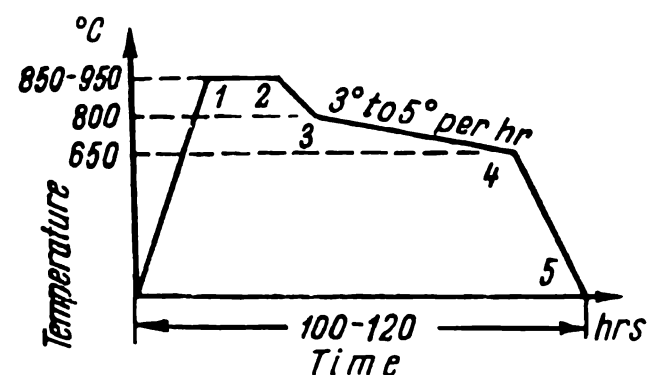


Fig. 78. Malleablising procedure for obtaining ferritic malleable iron

the metal is not decarburised. Ferritic malleable cast iron has a velvety-black fracture.

Either compartment or continuous tunnel furnaces are employed in making ferritic (black heart) malleable cast iron. The malleablising process is carried out in two stages of graphitisation. The first stage involves slow heating (line 0-1 in Fig. 78) to 850°-950° C and prolonged holding at this temperature (line 1-2).

Before being heated the cast iron has a structure of pearlite plus free cementite. During heating to point 1 the pearlite is transformed into austenite and a part of the secondary cementite dissolves in the austenite. Thus, at point 1 the cast iron consists of austenite with inclusions of structurally free cementite.

During the holding time (line 1-2) of 20 to 30 hours the free cementite is decomposed and temper carbon precipitates according to the reaction  $\text{Fe}_3\text{C} \rightarrow 3\text{Fe} + \text{C}$  (first stage of graphitisation). Two phases result from this process: austenite and temper carbon.

Section 2-3 is the intermediate stage of graphitisation during which a certain amount of secondary cementite is precipitated from the austenite and is graphitised.

The second stage of graphitisation occurs on line 3-4; it begins at  $700^\circ\text{--}750^\circ\text{C}$  with the decomposition of the cementite in the pearlite into ferrite and temper carbon. This process takes place at constant temperature or at a slow drop in temperature at a rate of 3 to 5 deg C per hour. The process ends at  $650^\circ\text{C}$ . Thus, at point 4 malleable iron is obtained having a structure of ferrite with inclusions of temper carbon. The structure of the metal does not change below point 4 and it is cooled outside of the furnace along line 4-5.

Castings of pearlitic malleable iron are classified into three grades according to their mechanical properties in which the tensile strength ranges from 30 to 40 kg per sq mm and the elongation, from 3 to 4 per cent.

Ferritic malleable cast iron is classified into grades on the same basis. The most widely used grades in the U.S.S.R. are K435-10 and K437-12. In the same manner as for high-strength cast irons, the first and second numbers are the tensile strength and the elongation, respectively.

It should be noted that even though these cast irons are called malleable they cannot be forged or wrought in any way because they are not capable of withstanding large deformations. This name was given to them because they possess higher ductility than ordinary grey cast irons.

**Alloy cast irons.** The addition of chromium, nickel, copper, titanium, molybdenum and other alloying elements increases the mechanical and physicochemical characteristics of the metal and, in many cases, enables castings with special properties to be produced.

In accordance with the alloying element or elements added the cast iron is known as nickel, chromium, chrome-nickel, etc., cast iron. Certain alloy cast irons are used to make parts subject to wear from friction. Parts of cast iron with a high chromium and nickel content possess anticorrosive properties and, in many cases, heat resistance.

**Cast steels.** In accordance with their chemical composition, cast steels are classified as:

- (1) plain-carbon,
- (2) low-alloy structural,
- (3) high-alloy with special properties, e. g., wear resistance, heat resistance, corrosion resistance in various aggressive media, etc.

The greatest quantity of steel castings is made of plain-carbon steel.

The most widely used grades of steel in Soviet foundries have the following chemical composition: 0.15 to 0.45% C, 0.5 to 1% Mn and 0.2 to 0.5% Si. The sulphur and phosphorus contents should be at a minimum (not exceeding 0.05 per cent). Steel castings of these grades have a ferrito-pearlitic structure and the following mechanical properties: tensile strength—not less than 40 to 60 kg per sq mm, yield point—from 20 to 35 kg per sq mm, elongation — from 10 to 24 per cent, reduction of area—from 18 to 35 per cent, and impact strength—not less than 2.5 to 5 kg-m per sq cm.

In comparison with cast iron, steel has a higher melting point, higher shrinkage (about 2 per cent), poorer fluidity and a higher susceptibility to the development of thermal stresses, and hot and cold cracks. To overcome these disadvantages much stricter requirements must be made to moulds for steel casting; they must have higher refractoriness and permeability as well as more strength and collapsibility than moulds for cast iron.

Almost all carbon steel castings undergo heat treatment to improve their microstructure and mechanical properties and to reduce the residual internal stresses.

Annealing, normalisation and hardening followed by tempering are treatments applied to steel castings. Internal stresses are relieved by annealing and by high-temperature tempering of carbon steels, involving slow heating to 650°-700° C followed by slow cooling (20 to 30 deg C per hour) to 400°-450° C.

Steel castings are used in many branches of industry.

#### 14-4. Casting Properties of Alloys

Proper filling of the mould cavity and the quality of the castings produced depend, not only upon the selected gating system, but upon the casting properties, or castability, of the metal as well.

The principal casting properties of metals and alloys are fluidity, susceptibility to segregation, and shrinkage.

*Fluidity* is the ability of the molten metal to completely fill and exactly reproduce the shape of the mould cavity. Metal with good fluidity will fill both thick and thin sections of a casting. Insufficient fluidity will lead to incomplete filling of the mould with metal thereby forming misruns, especially in the thinner sections of the casting.

Higher temperatures increase the fluidity of all metals and alloys. Nonmetallic impurities in the metal reduce its fluidity and impede its flow in the mould. The fluidity of cast iron is also reduced upon increasing the sulphur, oxygen and chromium content. Phosphorus,

carbon, silicon, manganese, aluminium and copper have the opposite effect—they increase the fluidity of cast iron.

The fluidity of metals is measured by making a processing test. The molten metal is poured into a cavity moulded in sand and having the form of a spiral of a definite cross section (Fig. 79). The length of the spiral is marked off in divisions, each 50 mm long. The fluidity value is the length that the molten metal flows before it solidifies in the spiral.

**Segregation**, i. e., chemical inhomogeneity, may occur in the solidification process of metals in moulds, and especially in large moulds. This inhomogeneity of the metal in various parts of the casting may lead to nonuniform mechanical properties.

**Shrinkage**, decrease in volume, is observed during cooling and freezing of the liquid metal in the mould. In practice, the term shrinkage, as used by the foundryman, means the difference between the dimensions of the pattern and those of the finished casting, expressed in per cent.

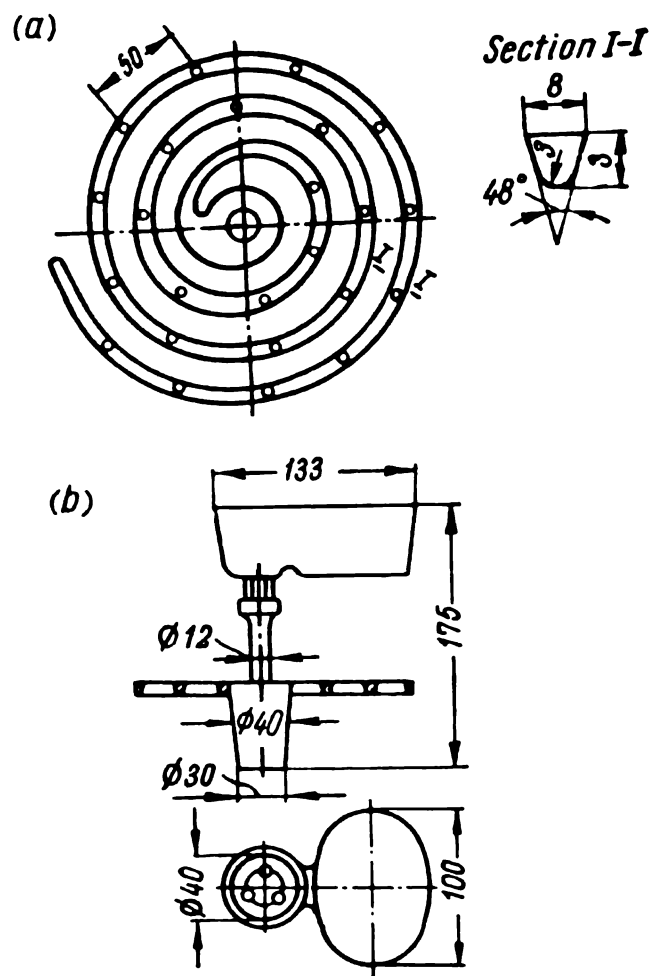


Fig. 79. Fluidity-spiral pattern for a fluidity test:

(a) spiral proper; (b) gating system

## 14-5. Pouring, Shakeout and Cleaning

**Pouring castings.** Metal, melted in a furnace, is tapped into a ladle from which it is poured into the moulds.

Ladles are available in a wide capacity range from 25-35 kg to 10 tons and even more

Small ladles are luted inside with clay while large ones are lined with refractory ladle brick or a refractory mass.

The lining must be thoroughly dried before filling the ladle with metal.

**Mould shakeout.** After a casting has solidified and cooled down sufficiently it is shaken from the mould. Prematurely shaken out castings may warp and therefore shakeout begins when the temperature of the casting does not exceed 400°-500° C.

Mould shakeout and core knockout are labour-consuming operations and are hazardous to the foundryman's health since a great deal of heat and dust are evolved. These operations are therefore mechanised in almost all foundries. Shakeout is usually done by means of vibratory knockouts, jolting grids and vibrators. In all cases, the mould is intensively jolted and broken up.

After shaking the casting out of the flask and removing the remaining moulding sand it is necessary to knock out the cores.

Cores are knocked out by means of stationary or portable core vibrators. It proves expedient to use air drills for knocking out the cores of heavy castings.

*Hydraulic core knockout*, hydroblasting, is a more sanitary process from the standpoint of dust suppression. It consists in breaking up and washing out the cores with a jet of water delivered at a pressure of 25 to 100 atm.

This process is carried out in steel cabinets. The casting enters the cabinet through a side door on a truck. This truck is run onto a motor-driven turntable. The operator, outside of the cabinet, directs a powerful jet of water on the casting through a nozzle.

Labour productivity can be increased several times by hydroblasting.

Recently, hydroblasting has been replaced by a process called hydrosandblasting which enables castings with a sufficiently clean surface to be obtained.

**Cleaning castings.** The next operation is called cleaning, or fettling, and involves the removal of adhering sand, the gating system and fins from the casting. It is done in the cleaning or fettling shop.

Surface cleaning to remove adhering sand is done in tumbling barrels and sand-blasting, pneumatic shot-blasting and airless shot-blasting machines.

Light castings may be cleaned in round or square tumbling barrels. The castings are loaded into the barrel with jack stars made of white cast iron. Rotation of the barrel causes the castings to tumble and abrade one another and thereby the adhering moulding sand is removed.

Sand-blasting and shot-blasting machines are widely employed for surface cleaning operations on light, medium and heavy castings. In these machines dry sand or cast iron shot is blown by a stream of compressed air against the surfaces of the casting. The impact of the sand or shot, travelling at a relatively high velocity, on the surface removes the adhering sand.

The most effective surface cleaning method at the present time is that using sand and water in a hydraulic cleaning plant. Here two operations are accomplished simultaneously: core knockout and surface cleaning.

A great variety of equipment is available for removing fins, risers, flow-offs, sprues and runners from castings and for finishing the casting surfaces by mechanical cut-off, torch cutting, electric-arc cutting, chipping, grinding, etc.

## *Chapter 15*

### SPECIAL CASTING AND MOULDING PROCESSES

Such progressive and up-to-date casting and moulding processes as permanent mould casting, centrifugal casting, die casting, investment casting, shell moulding and continuous casting have found wide application in the U.S.S.R.

#### **15-1. Permanent Mould Casting**

Among the numerous advantages of casting in permanent metal moulds are a sharp increase in labour productivity, lower percentage of rejects, higher mechanical strength of the castings, improved surface finish of the castings and the possibility of assigning smaller machining allowances.

While a sand mould serves but once each permanent metal mould can be used to make several hundred thousand castings of low melting point alloys, or from 1500 to 5000 grey iron castings, or from 600 to 700 small steel castings.

Permanent moulds are usually made in several pieces assembled together by fitting and clamping. They may be made of cast iron or steel. Mould life is extended by coating the surface of the mould cavity, with which the molten metal comes into contact, with a refractory lute. The coating is applied with a spray gun or brush after heating the mould to 200°-250° C.

#### **15-2. Centrifugal Casting**

Parts which have the shape of solids of revolution are most conveniently cast by the centrifugal method.

True centrifugal castings are made by pouring molten metal into the cavity of a rapidly rotating metal mould to whose walls the metal is thrown by centrifugal force and where it solidifies in the form of a hollow casting.

The centrifugal casting machines used to spin the mould may have either a horizontal or a vertical axis of rotation. Short castings

such as used for bronze bushings, worm wheels and piston ring blanks are usually cast with the spinning axis vertical. Long parts, as for example cast-iron water supply and sewerage pipes, steel gun barrels and others are cast in horizontal-type centrifugal machines.

Centrifugal castings acquire high density and are distinguished for their fine-grained structure and high mechanical strength. The absence of downsprues and flow-offs, the small machining allowances that can be assigned, the small percentage of rejects and the high output make centrifugal casting very economical.

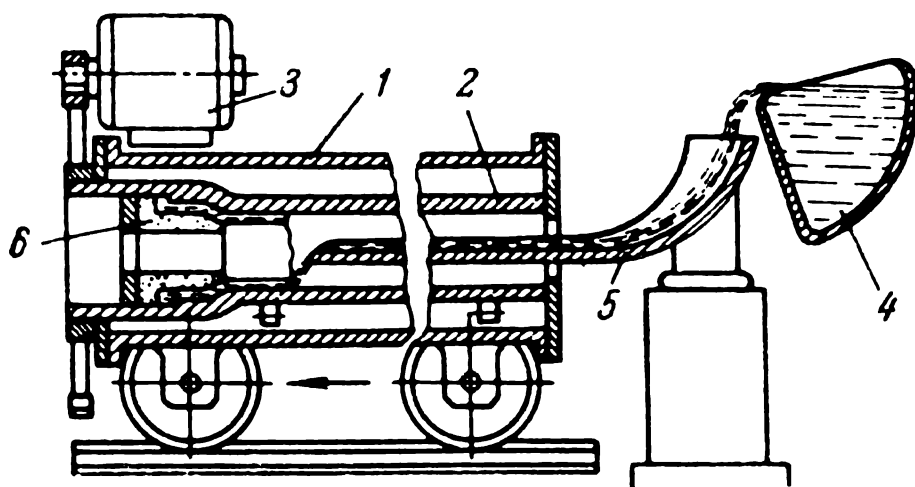


Fig. 80. Schematic diagram of a centrifugal pipe-casting machine:

1—shell; 2—spinning mould; 3—drive motor; 4—ladle; 5—trough;  
6—sand head core

Several types of centrifugal casting machines are available; their design is dictated by the size of the castings accommodated and their purpose. The schematic diagram in Fig. 80, for example, shows a machine for casting grey-iron water supply and sewerage pipes by means of this process.

There are two types of pipe-casting machines which differ in the way the metal is distributed along the length of the mould during pouring. In one the pouring trough travels horizontally, while the spinning mould is stationary; in the other the trough is stationary and the mould travels.

The spinning speed of the mould depends on the diameter of the pipe, the composition of the poured metal and a number of other factors.

Centrifugal casting of more intricate shapes (centrifuging) has been practised widely in recent years.

A foundry process devised to economise on nonferrous metals is the manufacture of two-layer (bimetallic) bushings in which the steel or cast iron bushing is faced with a layer of bronze. The process involves pouring molten grey iron into a spinning mould and, when it solidifies, pouring molten bronze into the same mould.

### 15-3. Die Casting

Die casting enables very accurate castings of complicated shapes to be produced. This process consists in forcing the molten metal under a certain pressure into the cavity of a steel mould, or die.

The principle of the submerged-plunger die-casting machine for aluminium alloys is illustrated in Fig. 81a. The molten metal is held in a steel pot 1, cast integral with cylinder 2. In the initial posi-

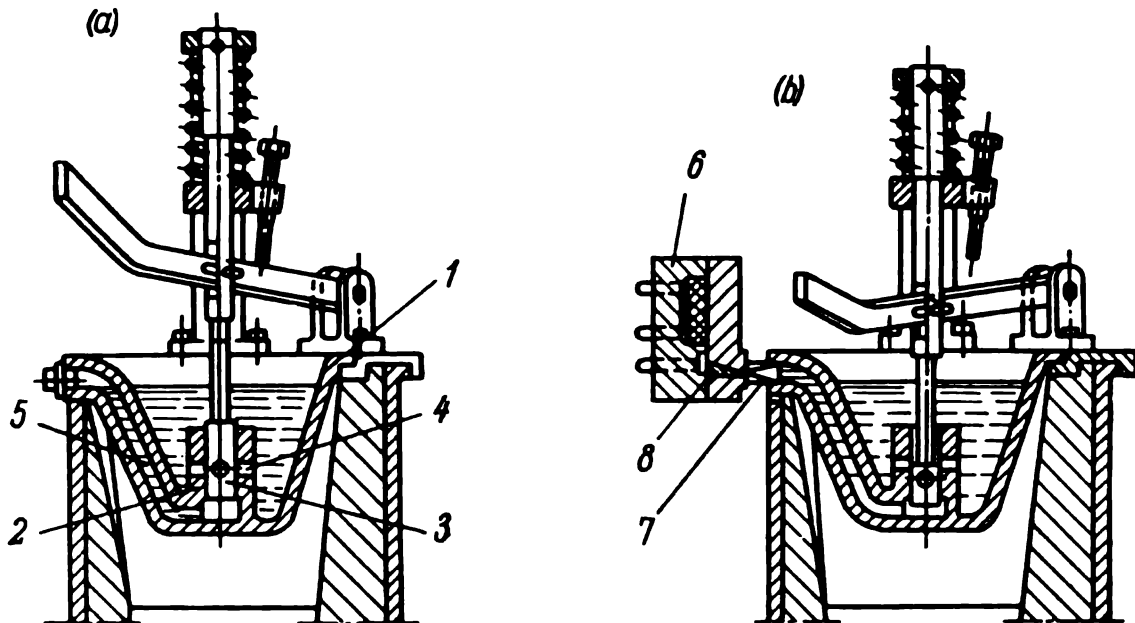


Fig. 81. Submerged-plunger die-casting machine:  
(a) principle of operation; (b) filling the die cavity with metal

tion of plunger 3 metal enters cylinder 2 through inlet 4 and is forced through gooseneck channel 5 into the die by the action of the plunger. Fig. 81b shows how die 6 is filled with liquid metal through nozzle 7 and gate 8.

At the present time parts of zinc-, aluminium-, magnesium- and copper-base alloys may be made by die casting.

### 15-4. Investment Casting

One of the problems raised by modern engineering is to produce castings of special steels and hard-cutting alloys with such high dimensional accuracy (within  $\pm 0.05$  mm) that they do not require subsequent machining. Such castings can be made by investment casting which is also known as the "lost-wax" process and "precision" casting.

The investment casting process comprises the following operations: (1) making a master pattern of the part to be cast, (2) making a composite die to the master pattern for casting the wax patterns, (3) casting the wax patterns in the die, (4) making the investment mould by



means of the wax patterns, (5) melting out the wax patterns and baking the mould, (6) melting the metal and pouring the mould, (7) shaking out the castings, removing gates and feeders and cleaning the castings.

The master pattern is usually made of a fusible alloy that can be easily machined. It is made oversize to compensate for wax and metal shrinkage. The composite die for casting the wax patterns is also made of a fusible metal and its cavity must be an exact replica of the master pattern. The assembled die is filled with molten wax which solidifies to form the wax pattern.

The gating system is also cast of wax in a die.

After extracting the wax pattern from the die it is joined to the pattern of the gating system and this wax assembly is coated with a slurry consisting of silica flour and small amounts of kaolin and graphite mixed in water with an addition of waterglass.

The wax assembly in the wet slurry coating is then dusted with a layer of calcinated silica sand which sticks to the surface and forms a quartz shell. This double coating on the pattern is air-dried for 5 or 6 hours on a rack and then invested in the mould.

A machine moulding process may be employed, and the moulding sand is a clay-bonded mixture (70 per cent silica sand and 30 per cent clay) with a moisture content up to 15 per cent.

The finished mould is dried in air for 2 or 3 hours and then baked in an oven for 2 hours to melt out the wax. At a temperature of 100°-120° C the wax melts and runs out through a hole in the bottom plate into a tray. When the wax is completely removed the flask is transferred to a drying furnace where it is first held at 150° and then gradually heated to 800°-900° C.

At this temperature the preheated mould is poured with molten metal which fills the cavity and solidifies. Investment castings produced by this process have a good surface finish and are exact reproductions of the master pattern.

### **15-5. Shell Moulding**

In some cases another progressive method of producing accurate castings, shell moulding, is resorted to.

This process consists in the following.

A metal pattern plate and the metal pattern secured on one of its sides are heated to 220°-260° C and coated with an emulsion to prevent the sand mixture from adhering to them. The moulding mixture, consisting of grains of silica sand (90 per cent) and thermosetting resin (6 per cent), is applied to the surface of plate and pattern thus prepared. First the resin in the layer of moulding mixture adjacent to the pattern melts and then it partially thermosets and, in 20 seconds,

builds up a homogeneous sand and resin shell from 5 to 8 mm thick on the pattern plate. Excess (unmelted) moulding mixture is removed and the pattern plate with the shell is placed into an oven where the shell is cured by holding at 250°-300° C for 1 to 3 minutes.

The solid shell formed on the pattern plate is a half mould. It is stripped from the pattern plate and assembled with the mating half shell by clamps or by pasting.

The finished shell mould is poured with molten metal through the gating system made together with the mould.

When the metal has been poured and cools sufficiently the mould is broken up and the sprues and risers are removed from the finished casting.

Semiautomatic shell moulding machines in use at the present time have an output of 60 and more moulds per hour per operator.

### **15-6. Production of Cast-iron Sheets by a Continuous Casting Process**

The principle of this process is that a controlled stream of molten grey iron is poured into the slit between two horizontal water-cooled plain mill rolls. In passing between the rolls the grey iron solidifies and forms a continuous strip from 0.7 to 1.2 mm thick. As a rule the strip leaves the rolls at a temperature from 1070° to 1090° C. Farther on the strip is cut into sheets which undergo heat treatment to eliminate brittleness.

Cast-iron sheets made by this method are used as a substitute for roofing iron.

## *Chapter 16*

### **QUALITY CONTROL IN THE FOUNDRY. CASTING DEFECTS AND SALVAGING OPERATIONS**

#### **16-1. Purpose and Methods of Quality Control**

The aims of quality control are to prevent defective products from being shipped from the foundry and to reduce the percentage of inevitable processing rejects.

Inspection procedures may be classed as follows:

(1) Visual examination of the castings, revealing such foundry defects as misruns, cracks, warping, etc. This is the most widely employed method, applicable both in piece and mass production of castings.

(2) Dimensional inspection of castings by means of measuring tools such as plug and snap gauges, template gauges and others to establish whether the dimensions of the casting conform to the drawing.

(3) Metallurgical control in which the chemical composition and the mechanical and other properties are determined in a laboratory.

## **16-2. Casting Defects and Salvaging Operations**

**Casting defects.** The main types of casting defects are:

(1) warping—lack of correspondence between the shape of the casting and the drawing due to bending and/or twisting of the whole or some part of the casting;

(2) burn-on—rough, irregular surface of the casting which cannot be cleaned by the conventional methods and is the result of metal penetration into the mould sand or melting of the latter or of a reaction between the metal and/or its oxides with the adjacent layer of moulding sand;

(3) hard spots—the presence of hard, unmachinable areas with a bright fracture at various parts of the casting;

(4) hot tears—through or surface cracks in the body of the casting, of straight or winding form, developed in the mould during cooling; the surface of the metal in the tear is usually oxidised;

(5) cold cracks—straight or winding ruptures and shrinkage tears in the casting formed as a result of nonuniform cooling and the development of internal stresses; the surface in such cracks is usually clean and unoxidised but sometimes temper colours are observed;

(6) cavities of various types—blows, shrinkage cavities, sand holes, etc.;

(7) shrinkage porosity—marked coarse-grain and unsound structure of the metal. It may be revealed by examining the machined surfaces of the casting or by hydraulic leakage tests. In many cases the shrinkage porosity is an accumulation of fine or even microscopic shrinkage cavities;

(8) mechanical damages—dents and nicks formed in shakeout, conveying or cleaning castings;

(9) nonconformity of the composition of the metal to the stipulations of the standards or specifications—a higher or lower content of certain chemical elements than required.

**Salvaging defective castings.** The most frequently used methods of repairing the defects of castings are: (1) liquid metal welding, (2) cold welding, (3) hot welding, (4) metal spraying, (5) luting and impregnation with sealing agents and (6) fitting plugs and bushings.

Liquid metal welding is resorted to when the cavity or crack on the surface of the casting is large. This treatment begins with cleaning the surfaces of the crack or cavity and moulding a cup or basin of clay

above it. Then the whole casting is preheated uniformly and molten metal is poured into the cup. Finally, the casting is slowly cooled.

Cold welding of cracks and cavities is permissible only on surfaces which are not to be subsequently machined. It may be done by means of either gas or arc welding. Long rods of cast iron, steel or copper, 5 or 6 mm in diameter, serve as the welding rod or electrode.

Hot welding is used for large cracks. In this salvaging process grey-iron castings are preheated to 500°-600° C and held at this temperature for 45 to 60 minutes. Only after this preparation are the castings welded with an oxy-acetylene flame using a welding rod of the following composition: from 3.2 to 3.5% C, from 3.5 to 4.0% Si and from 0.5 to 0.6% Mn. The flux to be used is a mixture of borax (50 per cent), soda (47 per cent) and silica sand (3 per cent). The welded part is annealed at 500°-600° C and is removed from the furnace at 50°-60° C.

Arc welding can be applied instead of gas welding.

Metal spraying (metallising), i. e., coating the surface to be repaired with a layer of atomised metal, is used to fill small cavities and shrinkage pores on unimportant surfaces of the casting which are not to be machined.

Luting and impregnation are employed to eliminate leakage in castings which operate under hydraulic pressure.

When the defects have been repaired the casting is inspected again in respect to its appearance, strength and pressure tightness.

## THE MECHANICAL WORKING OF METALS

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### *Chapter 17*

#### MECHANICAL WORKING PROCESSES. PREHEATING THE METAL

##### **17-1. Physical Nature of Plastic Deformation in Metals**

The shaping of metal by mechanical means in either the hot or cold state is called mechanical working. In mechanical working processes, the metal is shaped by pressure. This excludes machining or grinding, in which metal is machined off, and the casting of molten metal into a mould. Mechanical working processes include rolling, drawing, extrusion, smith forging, die forging and pressworking.

The initial materials for the mechanical working of metals are ingots and billets of various cross sections and weights.

**Effect of plastic deformation on the structure and properties of metals.** Mechanical working involves the plastic deformation of either cold or heated metal by the external action of special tools such as mill rolls, forging dies, flat hammer dies or extruding dies. Mechanical working changes the shape and dimensions of the billet but the volume of metal remains substantially constant throughout the process.

External forces acting on the billet in mechanical working lead to essential changes in the structure and properties of the metal since the size and shape of the grains are altered. Grain deformation in the metal is directly related to the way in which the billet is deformed and certain other factors. There are many reasons, such as the nonuniform size and mechanical strength of the grains, various directions and magnitudes of the acting forces, inhomogeneity of the stressed state throughout the volume of the deformed body, etc., why one group of grains is deformed less and another group more intensively, why a part of the grains is destroyed (split into smaller pieces), why the coarse-grained structure of the ingot is refined and

why only elastic stresses are developed in the crystal lattices of certain grains.

The deformation of metal when it is worked mechanically in the cold state (cold working) leads to strain or work hardening. Work hardening is associated, not only with an increase in the tensile strength and hardness of the metal, but with an appreciable decrease in plastic properties (elongation, impact strength, etc.). In work hardening the grains are stretched in the direction of tensile deformation thereby leading to anisotropy (directional properties) of the metal. After heavy work hardening further deformation of the metal in the cold state may become entirely impossible due to the development of discontinuities (cracks, tears and other defects that occur in the metal).

If metal is heated to a certain definite temperature before subjecting it to mechanical working, there is less danger of nonuniform deformation of the grains and the consequent development of stresses since the latter are practically eliminated by the recrystallisation process that proceeds simultaneously with deformation.

Recrystallisation denotes the formation and growth of new grains of metal from the fragments of the just deformed grains. As a result, the crystal lattice of the metal, distorted by mechanical working, is restored and the elastic strain in the metal is eliminated.

**Effect of various factors in the mechanical working process on the plasticity of metals.** Metals and alloys of different nature do not have the same ductility and, therefore, behave differently when subjected to hot and cold working.

It has been established that the plasticity of a metal depends upon its chemical composition, structure (phase content and grain size), heating temperature, rate and degree of deformation, and the direction and magnitude of the external forces acting on the metal (system of stressed state and system of deformation).

It is a fact that pure metals have a higher plasticity than their alloys since in the latter new structural components, including chemical compounds of the basic elements, may be formed. Such compounds may materially change the plasticity of the basic metal. For example, steel with a low carbon content has higher plasticity than steel with a higher content; pure copper is many times more plastic than its alloys with tin (bronzes), etc.

The plasticity of a cast metal having a coarse-grained dendritic structure proves to be less than that of a metal having undergone deformation and having, therefore, a more fine-grained structure. If an annealed metal in which the structural phases are in an equilibrium state is subjected to deformation, its ductility will be much higher than that of the same metal after a hardening in which a non-equilibrium structure is retained.

As a rule, the ductility of a metal increases with temperature, and its resistance to deformation is reduced.

The rate of deformation in mechanical working affects the plasticity of the metal in various ways. An increase in the deformation rate within a definite limit is accompanied by a decrease in plasticity. A further increase in this rate, after a certain point, improves the plasticity.

The degree of deformation, both in hot and, especially, in cold working, must be maintained within known limits. Rates exceeding the upper limit lead to the formation of cracks, tears and other discontinuities in the metal.

The type of stressed state and the type of deformation also affect the ductility of the metal in the course of a mechanical working operation.

In the various mechanical working processes there are usually two different stressed state systems: (1) three-dimensional compression of the metal when compressive stresses act along all the three axes, as in smith forging, closed-die forging, rolling and extrusion; and (2) three-dimensional stressed state of the metal when compressive stresses act along two axes and a tensile stress along the third, as in drawing.

Metal has a higher ductility in the first system of stressed state than in the second.

The three *systems of deformation* to which metal is most frequently subjected in mechanical working are (1) compressive strain along one axis and tensile strain along the two others; (2) compressive strain along the first axis, tensile along the second and none along the third; and (3) compressive strain along two axes and tensile strain along the third. The less the role of tensile stresses in these systems, the higher the capacity for plastic deformation that the metal will display. One and the same metal may have good ductility when worked by the third system of deformation and poor ductility when the first is applied.

## 17-2. Preheating the Metal

**Mechanical working temperature ranges.** Each metal has a definite temperature range within which mechanical working can be done to the best advantage, a range where good plasticity of the metal is ensured in conjunction with its minimum resistance to deformation.

The basis for properly selecting the mechanical working temperature is the equilibrium diagram of the metal or alloy. The minimum preheating temperature for hypoeutectoid carbon steels is from 50 to 100 deg C above the line GS and for hypereutectoid steels, 50 to 100 deg C above the line SK (Fig. 82). The maximum preheating temperature is from 150 to 200 deg C below the solidus. The shaded area shown in Fig. 82 is the temperature range for working carbon steels.

Before undergoing deformation ingots or billets should be uniformly heated throughout their cross section without overheating or burning the metal; this requirement must be especially strictly observed in the working of steel.

Cracks and fissures may be formed in overheated metal in the process of working.

In many cases, the structure of forged or rolled metal obtained upon overheating may be corrected by subsequent heat treatment; the burning of the metal in forging or rolling operations leads to unsalvageable spoilage.

Steel ingots are most frequently preheated for rolling in soaking pits; billets are preheated in continuous and compartment furnaces.

**Soaking pits.** Two types of soaking pits, regenerative and recuperative, are used at present.

In *regenerative soaking pits* the heat of the outgoing combustion products is utilised to preheat the fuel gas and air in the regenerators before they burn in the pit.

The design of the regenerators in a soaking pit has much in common with that of open-hearth regenerators. The hot waste gases first pass through the refractory checkerwork of one pair of regenerators and heat it to a high temperature. Then the valves are reversed and cold gas and air are admitted separately into the heated chambers of the regenerator where they are preheated by contact with hot checkerwork and pass further to the burners.

The regenerators operate periodically: while cold gas and air are being heated in the checkerwork of one pair of regenerators, the checkerwork of the other pair is being heated by the outgoing products of combustion.

*Recuperative soaking pits* utilise the heat of the outgoing gases for continuously heating the incoming air through the walls of the recuperators which consist of a row of cast-iron, steel, or ceramic pipes. The hot products of combustion pass through these pipes and give up their heat to the cold currents of air flowing along the external surfaces of the pipes.

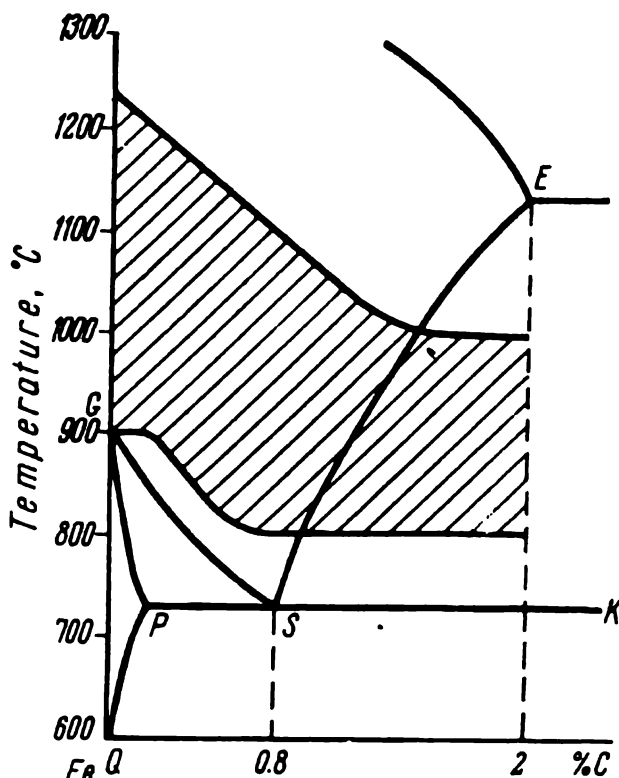


Fig. 82. Temperature range for the mechanical working of carbon steel



A cross section of one unit of a recuperative soaking pit for heating ingots is shown in Fig. 83. This pit is fired from the central part of the pit bottom. Usually, a group of soaking pits is provided in a rolling department. Each pit consists of two or more units accommodating from 12 to 22 ingots.

To charge the ingots into the pit, cover 1 is removed by means of a floor crane. After charging, the cover is replaced and blast-furnace or coke-oven gas and air preheated in the recuperators are delivered to the central burner 2. The flame produced in combustion passes

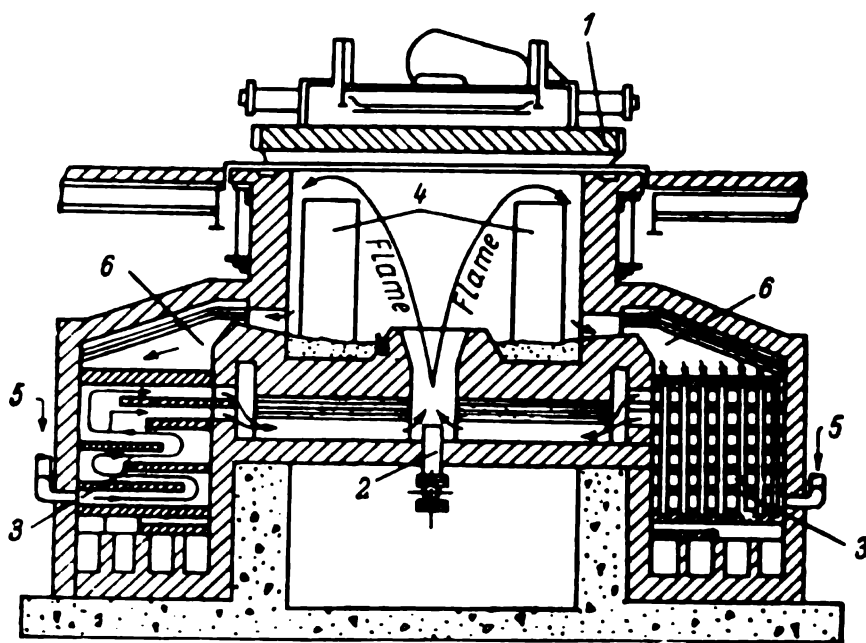


Fig. 83. Recuperative soaking pit

over the ingots 4, heating them, while the products of combustion (waste gases) pass through the side channels 6-6 into recuperators 3-3. Pipelines 5-5 serve to deliver cold air into the recuperators.

The operation of modern soaking pits is fully automatic; the gaseous fuel and air flow is automatically turned on and off, the pressure of the gases in the units is controlled automatically and the valves are reversed in the same manner.

Ingots, heated to the required temperature, are drawn from the pit with an ingot crane which deposits them on an ingot car. The latter delivers them to the receiving table of the rolling mill.

**Continuous furnaces.** Fig. 84 illustrates a typical design of a continuous furnace for preheating billets before rolling or forging.

In continuous furnaces the metal being heated travels in a direction opposite to the flow of the hot gases. The billets are charged through the end door by pusher 1 which pushes the row of billets along two skids into the working zone of the furnace while the products of combustion flow in the opposite direction (from right to left). The cold billets enter the preheating zone 6 of the furnace which has a temperature of

500°-600° C at the beginning and 900°-1000° C at the end. Having passed through this zone the billets enter the soaking zone 7 where the temperature is 1250°-1300° C. The temperature in this zone is maintained from 50 to 100 deg C higher than the temperature of the metal before rolling. The heated billets are discharged to table 2 and conveyed to the rolling mill, forging hammer or press.

The hot gases flow from the end 3 and bottom 4 burners in the face of the advancing billets and, after giving up a part of their heat, pass through the vertical channels in recuperators 5 from where they are discharged through the flue and stack.

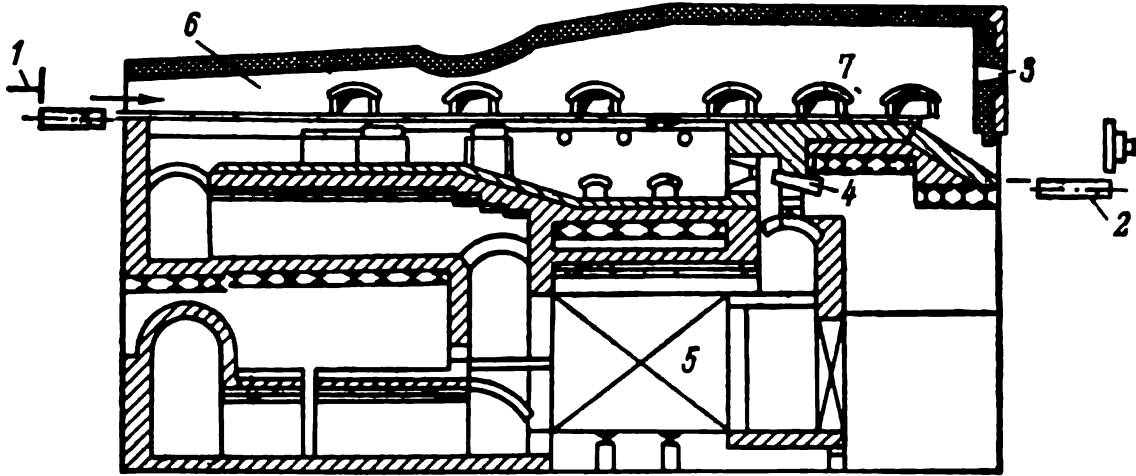


Fig. 84. Continuous preheating furnace

Some continuous heating furnaces incorporate a soaking zone where the metal is held to bring it to a more uniform temperature throughout its cross section.

Other types of furnaces employed for preheating metal for hot working are car-bottom, rotary-hearth, walking-beam-bottom conveyer, roller-bottom, cover-type and electric furnaces.

## Chapter 18

### FUNDAMENTAL PRINCIPLES OF THE METAL ROLLING PROCESS. ROLLING MILL DESIGN AND CLASSIFICATION

#### 18-1. Fundamental Principles of the Metal Rolling Process

The process of shaping metals and alloys into the semifinished or finished condition by passage between rolls is known as rolling. The rolls rotate in opposite directions, one counter-clockwise and the other clockwise. The rolling of strip metal is schematically shown

in Fig. 85. Rolling involves plastic deformation of the metal in which the thickness of the strip is reduced and the length and width are increased.

The difference between the initial and final thicknesses of the strip is called the absolute draught:

$$\Delta h = (h_0 - h_1) \text{ mm.} \quad (68)$$

The difference between the final and initial lengths of the strip is called the absolute elongation:

$$\Delta l = (l_1 - l_0) \text{ mm.} \quad (69)$$

The difference between the final and initial widths of the strip is called the absolute spread:

$$\Delta b = (b_1 - b_0) \text{ mm.} \quad (70)$$

The deformation of the strip in rolling can also be characterised by:

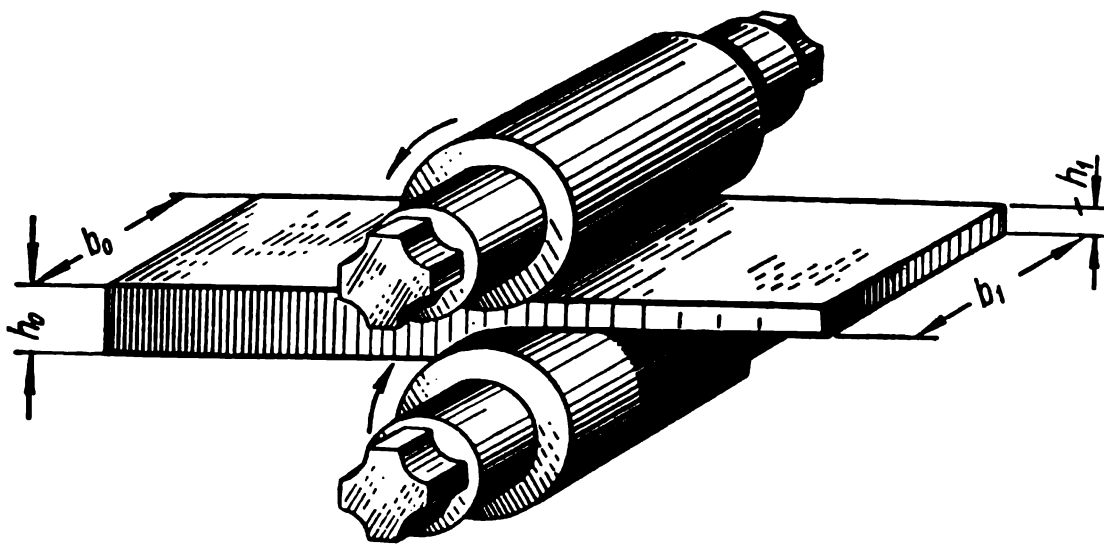


Fig. 85. The metal rolling process

(1) the relative draught, i.e., the ratio of the absolute draught to the initial thickness of the strip, expressed in per cent:

$$\varepsilon = \frac{\Delta h}{h_0} 100 = \frac{h_0 - h_1}{h_0} 100, \quad (71)$$

(2) the coefficient of elongation, i.e., the ratio of the length of the strip after rolling to its initial length:

$$\mu = \frac{l_1}{l_0}. \quad (72)$$

Since the volume of the metal is not changed by rolling, then  $h_0 b_0 l_0 = h_1 b_1 l_1$ . It follows that

$$\mu = \frac{h_0 b_0}{h_1 b_1} = \frac{F_0}{F_1}. \quad (73)$$

Thus, the length of the strip is increased in rolling proportional to the reduction of its cross-sectional area. Under ordinary conditions, the main result of the reduction of the metal in thickness is the increase in length since the spread is comparatively small.

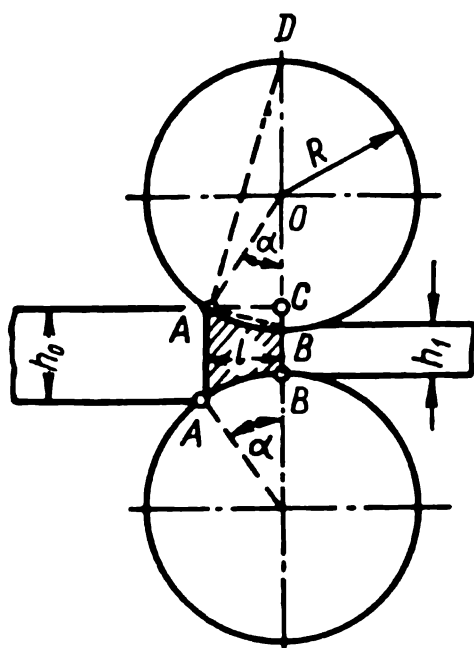


Fig. 86. Zone of deformation and angle of contact in rolling

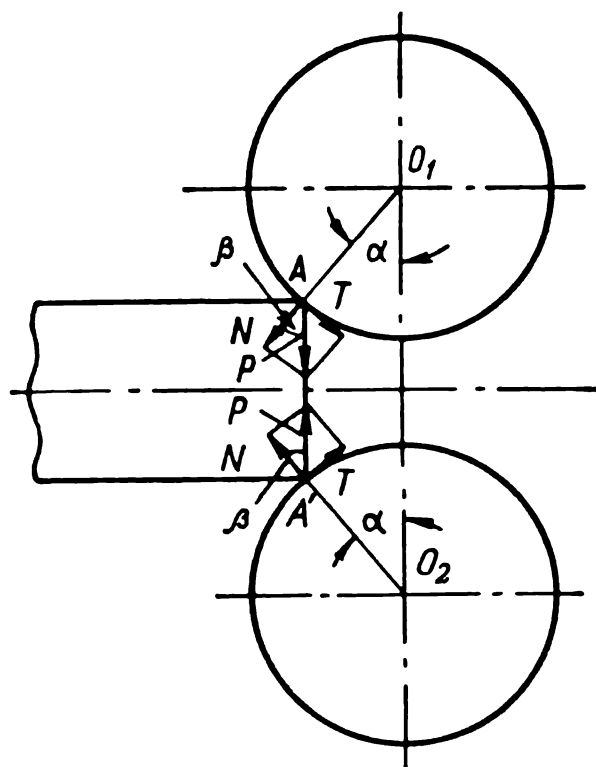


Fig. 87. Forces involved at the point of entrance of a bar between the rolls

**The action of forces on the metal in rolling.** The metal contacts each of the rolls along the arc  $AB$  (Fig. 86) which is known as the *arc of contact*. This arc corresponds to the central angle  $\alpha$ , called the *angle of contact* or *bite*.

The volume of metal bounded above and below by the arcs of contact  $AB$ , by the side edges of the strip and by the planes of entrance and exit of the metal from the rolls is called the *zone of deformation* of the metal.

The angle of contact is determined by the formula:

$$\cos \alpha = 1 - \frac{h_0 - h_1}{2R}. \quad (74)$$

This formula shows the relation between the angle of contact  $\alpha$ , draught  $\Delta h = h_0 - h_1$  and the roll diameter  $D = 2R$ .

The process of metal rolling is made possible by the friction that occurs between the contact surfaces of the rolls and the strip or bar being rolled.

At the moment of bite (entry between the rolls) two forces act on the metal from the side of each roll (Fig. 87). They are the normal (radial) force  $N$  and the tangential force  $T$ . We know from mechanics that in the relative motion of two bodies the friction force developed is equal to the normal force multiplied by the coefficient of friction:

$$T = Nf \quad (75)$$

and that the ratio of the friction force to the normal force equals the tangent of the angle of friction  $\beta$ , i.e.,  $\frac{T}{N} = \tan \beta = f$ .

To ensure entry of the bar between the rolls it is necessary that the coefficient of friction be larger than the tangent of the angle of friction,  $f > \tan \beta$ , or that the angle of contact be less than the angle of friction  $\alpha < \beta$ .

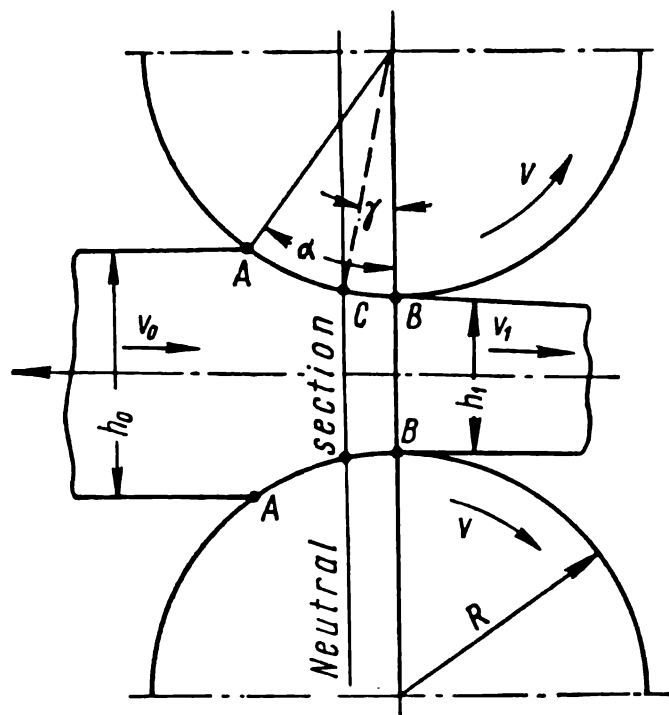


Fig. 88. Location of the neutral section in the zone of deformation

The maximum permissible angle of bite (contact) depends upon the materials of the rolls and the bar being rolled, the roughness of their surfaces, and the rolling temperature and speed. As a rule, the maximum angle of bite ranges from  $24^\circ$  to  $30^\circ$  in rolling blooms and heavy billets, from  $15^\circ$  to  $20^\circ$  in hot rolling sheet and strip stock, and from  $2^\circ$  to  $10^\circ$  in cold rolling oiled sheet and strip.

It has been established both theoretically and experimentally that the speed  $v_1$  at which the bar is delivered by the

rolls is somewhat higher, and the bar entrance speed  $v_0$  is somewhat lower than peripheral speed  $v$  of the rolls, i.e.,  $v_1 > v > v_0$  (Fig. 88).

There is a cross section in the zone of deformation in which the speed of the metal  $v_m$  is equal to the peripheral speed  $v$  of the rolls. This section, in which the horizontal component of the roll speed is equal to the speed of the metal, is called the nonslip (or critical) section; point C on the arc of contact, at which  $v_m = v \cos \gamma$ , is called the nonslip point and the corresponding central angle  $\gamma$  is the nonslip (critical) angle.

To the left of the nonslip section the metal travels with less speed than the peripheral speed of the rolls; this part of the zone of deformation is called the *lagging zone*. To the right of the neutral section the metal travels faster than the peripheral speed of the rolls, i.e., it leads the rolls, and this part of the zone of deformation is called the *forward slip zone*.

The value by which the speed of the bar leaving the rolls exceeds their peripheral speed is known as the *forward slip*. It is determined as the ratio of the difference between the bar delivery speed and the peripheral speed of the rolls to the peripheral speed of the rolls, and is expressed in per cent:

$$S = \frac{v_1 - v}{v} 100, \quad (76)$$

where  $v_1$  = speed of metal delivery from the rolls,  
 $v$  = peripheral speed of the rolls.

Forward slip increases with the radius of the roll and with a decrease in the thickness of the bar; it increases with an increase of the coefficient of friction and may vary from 3 to 10 per cent.

When metal is reduced between rolls, a certain increase in width of the bar may occur in addition to elongation. From the condition that the volume of metal remains constant in rolling, it follows that for a given draught the elongation and spread of the bar are interrelated:

$$\frac{h_0}{h_1} = \frac{l_1}{l_0} \cdot \frac{b_1}{b_0}.$$

Upon constant reduction in height, the less the elongation, the more the spread will be and vice versa. The ratio between elongation and spread, when the bar is rolled with a certain definite draught, depends upon the ratio between the corresponding principal stresses acting on the metal in the zone of deformation.

The main stress system in the metal in rolling is triaxial compression (Fig. 89). Under ordinary rolling conditions the principal stress  $\sigma_1$  along the height has the maximum value, the longitudinal principal stress  $\sigma_3$  has the minimum and the lateral principal stress  $\sigma_2$ , a medium value. Consequently, longitudinal deformation (elongation) prevails over lateral deformation (spread).

Spread increases with an increase in roll diameter and the coefficient of friction, as well as with a fall in temperature of the metal in the course of hot rolling. Spread is proportional to the draught and depends upon the thickness and width of the bar.

In calculations of the strength of the rolls and other parts of a roll stand, and also in determining the required power of the mill drive motors it is necessary to know the full roll load, or load separating force.

To determine the full roll load  $P$  it is first necessary to find the horizontal projection of the area of contact  $F$  between the metal and the rolls and the mean unit pressure  $P_m$  on this area. Then, the full load is found from the formula:

$$P = P_m F. \quad (77)$$

In rolling simple shapes—sheets, strips and billets of rectangular or square cross section—the projected contact area is determined as the product of the mean width of the bar in the zone of deformation

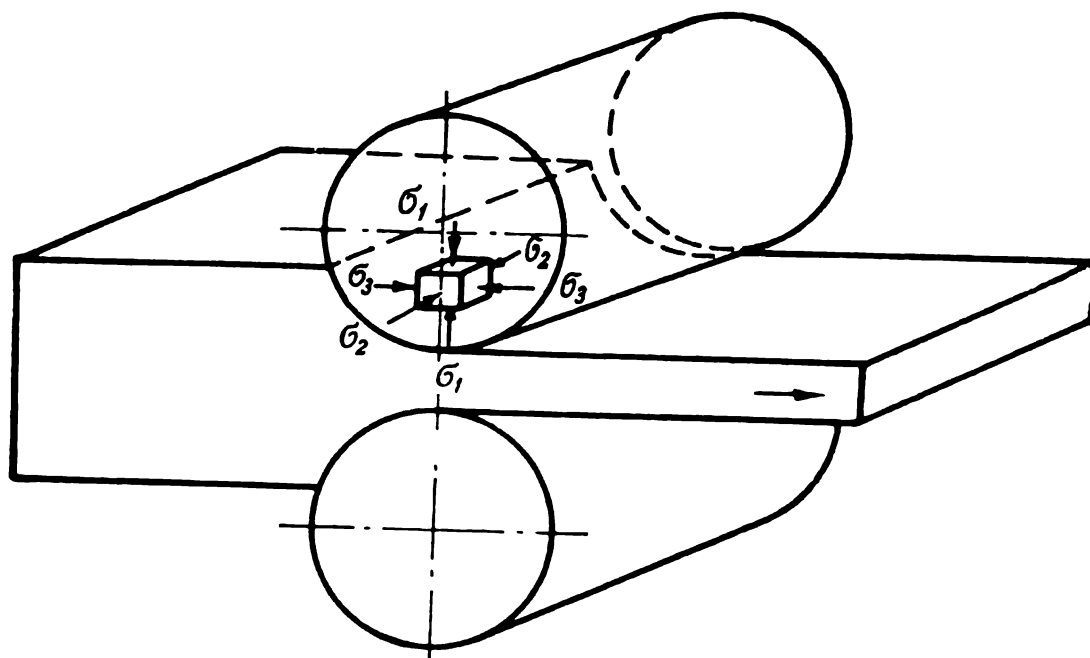


Fig. 89. The stressed state in rolling

and the length of this zone. In rolling complex shapes—angles, channels, beams, rails, etc.—the projected contact area is determined graphically or from approximate formulas. The mean unit pressure of the metal on the rolls is determined theoretically from plasticity equations or is taken from experimental or practical data.

The required power rating of the mill motors can be found by first determining the torque required for rolling. The motor torque for mills with a constant rolling speed during the whole time the metal passes through the rolls is the sum of the moment of forces required to deform the metal and the moment of frictional forces. The first addend is the torque required to deform the metal in the rolls; the second is the moment of frictional forces appearing in the roll bearings and the gearing of the transmitting mechanisms. In mills with variable rolling speed, it is also necessary to take into consideration the torque required to accelerate the motor when the rolling speed is increased. This is called the dynamic torque.

**Nonuniform deformation of the metal in rolling.** Rolling frequently involves nonuniform deformation (reduction) of the metal.

The simplest case of nonuniform draught is the flattening of a round bar between plain-barrelled rolls. In this case the draught varies along the width of the bar; nonuniform deformation is caused by the shape of the cross section and nonuniform heating across the bar section. Another reason may be provided by the grooves in the rolls for changing the shape and size of the cross section of the bar.

Nonuniform reduction is most pronounced in the rolling of complex shapes, such as rails, beams, channels, grooved piles, etc.

In nonuniform reduction, various parts of the cross section tend to elongate differently depending upon their draught. However, due to the influence of the rigid bar ends and the cohesion between the particles of metal, a general (mean) elongation of the metal is established across the whole cross section of the bar. This constrained equalisation of the elongation may be accompanied by local changes in the lateral dimensions of the bar in the form of pulling-down or induced spread of the metal, rupture or the concentration of internal stresses in the metal. All these factors must be given due consideration in the production of complex rolled shapes.

**Roll pass design.** Steel plate, sheet and strip are rolled between rolls having a smooth, cylindrical; slightly cambered (convex) or concave barrel (working surface). Bars, rods and special-purpose shapes are rolled between grooved rolls, i.e., rolls in which grooves have been cut corresponding to some extent to the finished shape of the bar being rolled. The shape cut into one roll is called the *groove*; the shape formed when the grooves of mating rolls are matched together is called the *pass*.

By rolling the metal consecutively through the passes, the initial square or rectangular cross section of the ingot (or billet) can be gradually changed to obtain a bar of the final required shape.

Passes designed for reducing the cross section of the bar are known as *breakdown*, or *roughing*, *passes*.

Breakdown passes may be of the rectangular (box), diamond, square and oval types (Fig. 90).

Box passes are employed in blooming mills, and in roughing and continuous billet mills. Rolling proceeds, as a rule, in such passes until the bar is reduced to a cross section of  $100 \times 100$  mm; the reduction varies from 10 to 30 per cent and the coefficient of elongation—from 1.1 to 1.25.

Diamond passes comprise a system, or sequence, of similar diamonds in which a bar of larger cross section is entered into each next smaller pass after turning it  $90^\circ$  about its axis. The coefficient of elongation of the metal when it is rolled in a diamond pass depends upon the shape of the diamond. The larger the ratio of the diagonals



(or the larger the obtuse angle), the greater the reduction that can be accomplished without danger of overfilling the pass with metal. Diamonds with obtuse angles from  $93^\circ$  to  $100^\circ$  are commonly employed; the respective coefficients of elongation vary from 1.2 to 1.4.

A sequence of passes in which diamonds alternate with squares is used chiefly in rolling square bars and flats. More stretched out

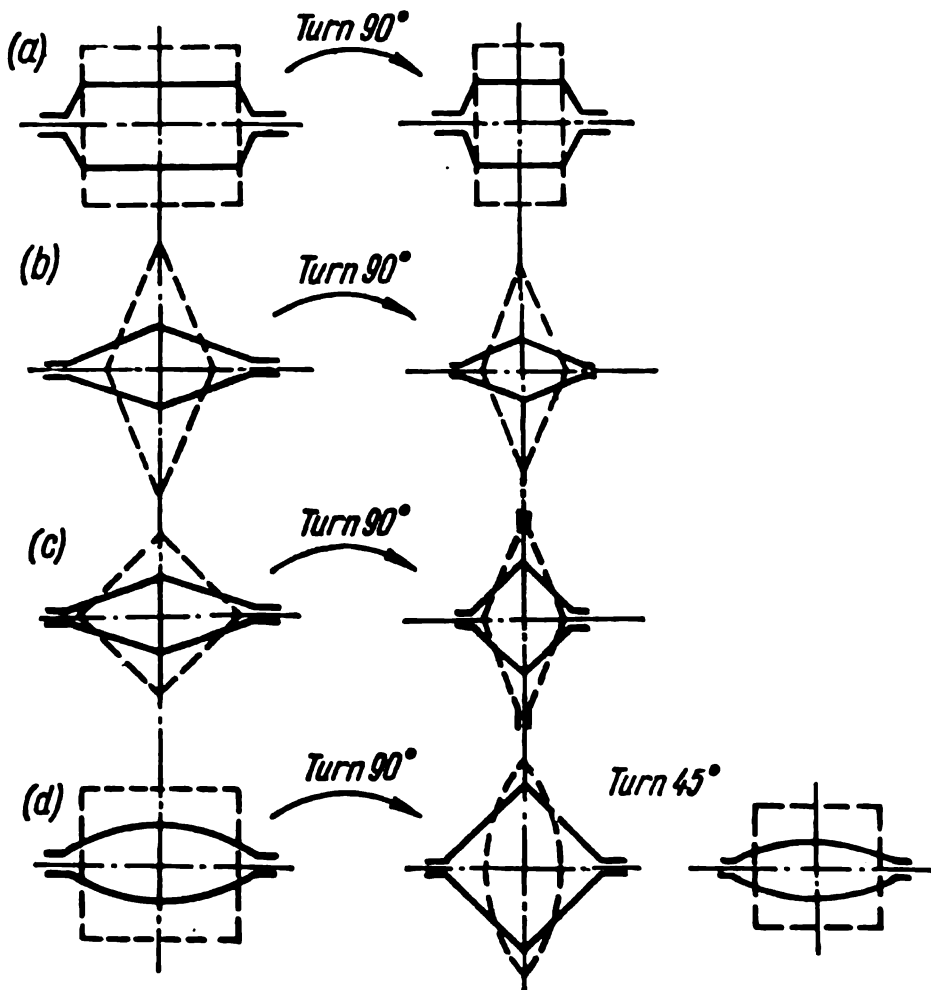


Fig. 90. Breakdown pass sequences:

(a) box passes; (b) diamond-diamond series; (c) diamond-square series;  
(d) oval-square series

diamonds, with obtuse angles from  $100^\circ$  to  $125^\circ$ , are employed in this sequence. The coefficient of elongation ranges from 1.25 to 1.5.

The most effective and widespread system of breakdown passes is the oval-square series. Reduction (elongation) in this case depends upon the shape of the oval; the more the oval is flattened (the higher the ratio of its axes), the higher the draught that can be used in rolling a square bar in the oval pass and also in entering the oval bar into the next square pass. In flattened ovals the ratio of the axes may equal two or more and the coefficient of elongation may be from 1.5

to 2.5 and even higher. Large elongation enables very small sections to be rolled and therefore the oval-square series is the main sequence used in producing wire-rod and other small sections.

*Leader passes* (Fig. 91) are next to the last in the course of rolling and serve to gradually bring the cross section of the bar to the final shape. A great variety of leader pass designs are employed; for example, the finishing pass for rolling rounds is preceded by an oval leader pass, the finishing pass for rolling squares is preceded by a diamond leader pass, etc.

The shape of the last or *finishing pass* should be identical to that of the finished rolled stock with due consideration for the linear coefficient of thermal expansion and for the dimensional tolerance of the final product. Moreover, a clearance between the rolls must be provided so that they can be brought closer together before rolling by an amount equal to the magnitude of the elastic deformations of the rolls and various parts of the stand. The sum of all these deformations is called roll spring and it tends to increase the distance between the rolls.

Spread is another factor that affects pass design. If space has not been provided in the pass for spread, the metal will be forced into the clearance between the rolls and fins will be formed on the bar. Fins may lead to various defects such as laps, cracks, etc.

Vertical side walls of a pass are always slightly inclined in reference to the roll axis. This avoids jamming of the bar in the pass and enables the pass to be restored to its initial size and shape after wear by a redressing operation (involving turning or grinding).

The most important job in roll pass design is to work out the draught schedule. This schedule must take account of the ductility of the metal and its resistance to deformation, permissible angle of bite, strength of the rolls and other parts of the mill, available power of the drive motor and the spread value. In designing passes for the rolling

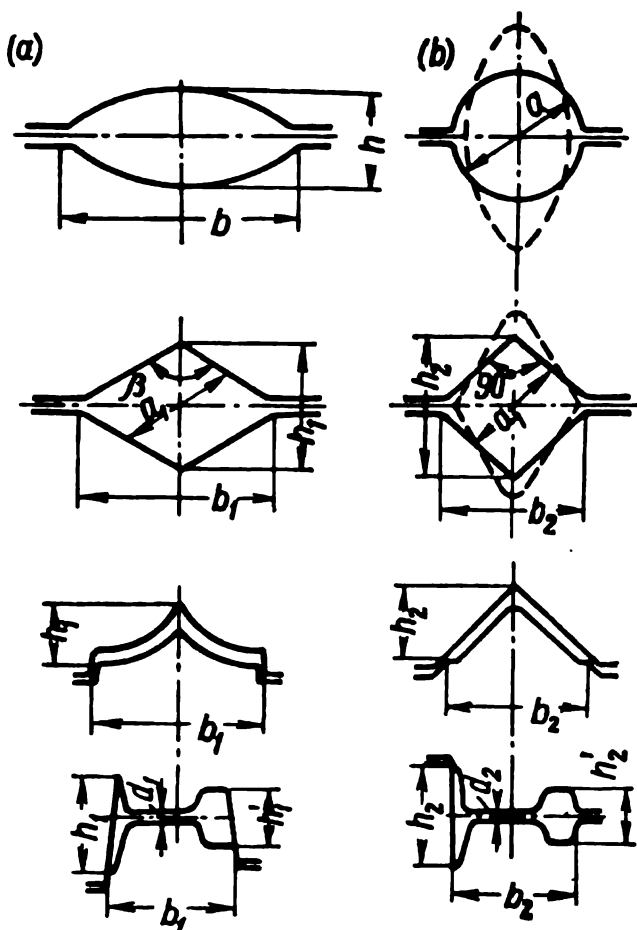


Fig. 91. Leader (a) and finishing (b) passes

of complex shapes, such as beams, channels and rails, consideration must also be given to factors associated with nonuniform deformation (pulling-down and induced spread).

## 18-2. Rolling Mill Design and Classification

A rolling mill consists of the following principal units: (1) one or several roll stands, (2) main drive motor, (3) reducing gear, (4) pinion stand, (5) flywheel and (6) coupling gear between the units (Fig. 92). The rolling mill is erected on a single foundation.

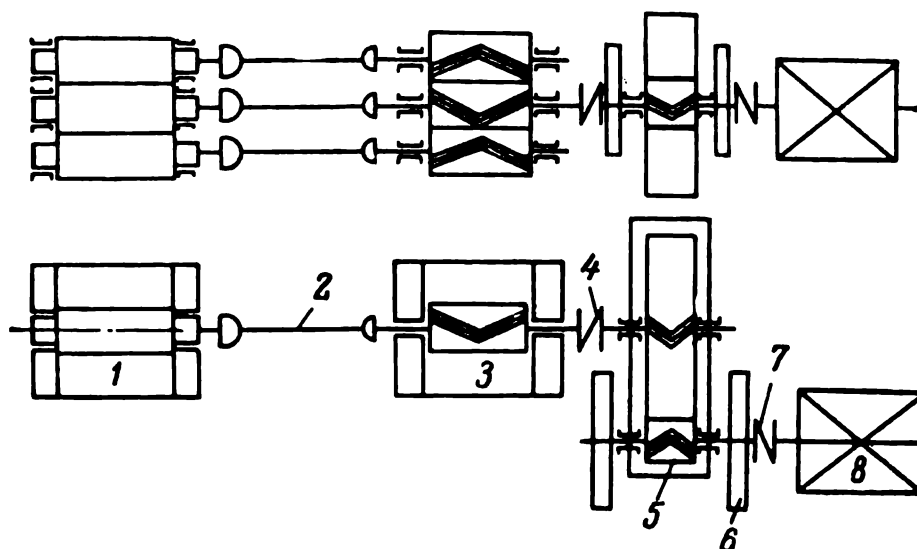


Fig. 92. Main line of a rolling mill:  
1—roll stand; 2—spindle; 3—pinion stand; 4—main coupling; 5—reducing gear; 6—flywheel; 7—motor coupling; 8—motor

**Roll stand.** The roll stand is the main unit of a rolling mill since in it the actual rolling of the metal is performed (Fig. 93). It consists of two housings 1 with feet 3; bed plates 2, on which the housings rest; separators 4, joining the housings; housing tops 5; working rolls 6; chocks 7, 8 and 9 of the bottom, middle and top rolls, respectively, with their bearings; roll setting or screwdown mechanism 10 and top roll balancing device 11.

*The housings* of roll stands mount the rolls and serve to carry the roll load which is transmitted through the roll necks and chocks. Plain or antifriction bearings are fitted into the chocks for the roll necks.

Housings may be either of the closed- or open-topped type. Closed-topped housings enable shapes of high accuracy to be rolled but roll changing is more difficult than in a stand of open-topped housings. Closed-topped housings are employed in heavy breakdown and sheet and plate mills, subject to a high rolling force (up to 2000 tons). In this case, rolls are changed through a window of the housing by

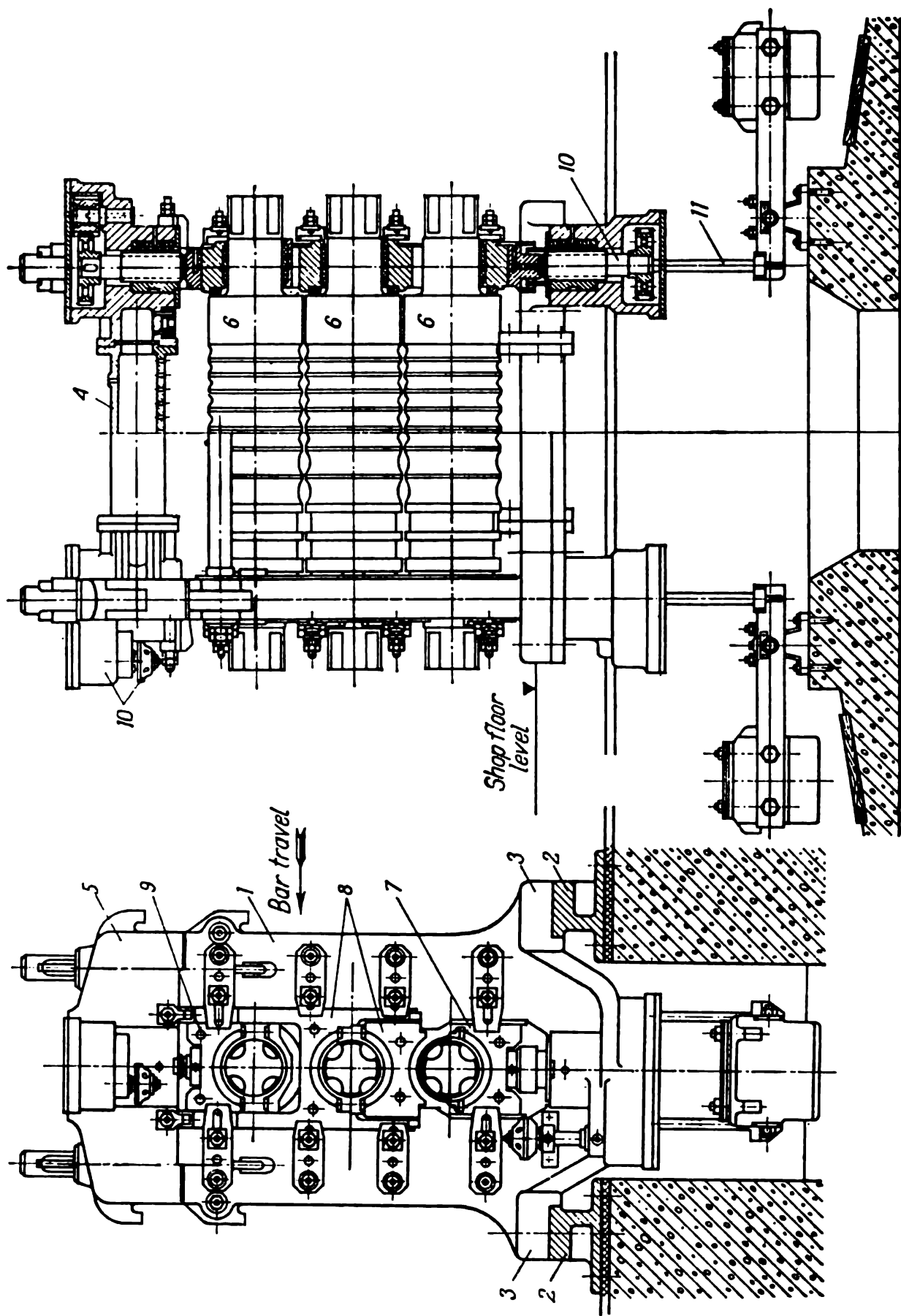


Fig. 93. 500-mm three-high roll stand of a heavy-section mill (of Soviet design)

means of a truck. Finishing roll stands also have closed-topped housings. Open-topped housings are used in breakdown and roughing stands in cases where the rigidity of the housings is not of vital importance.

The rolls perform the actual operation of reducing the metal and imparting to it the required shape. In the process of deforming the metal the rolls are subjected to the pressure exerted by the metal and transmit this pressure to the bearings.

A mill roll (Fig. 94) comprises the following elements: barrel 1, which directly contacts the metal in rolling; roll necks 2, on both ends of the roll barrel and supported in the bearings; and wobble necks 3, which serve to couple the roll to the spindle.

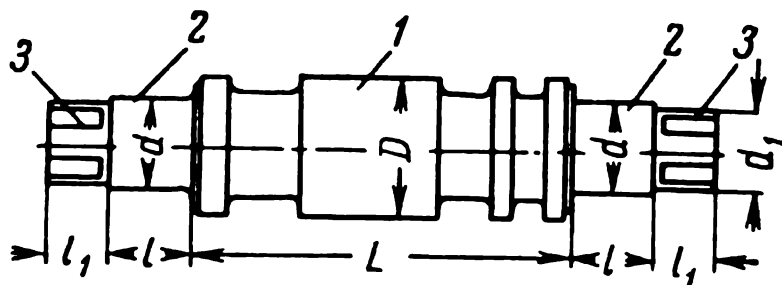


Fig. 94. Elements of a working roll

Rolls are made of either cast iron or steel.

Soft grey-iron rolls, cast in sand moulds, are employed in roughing passes for the hot rolling of steel. Hard grey-iron rolls, cast in metallic moulds, have a chilled surface layer and find widespread application in sheet and plate mills, and in the finishing stands of section and rod mills.

Cast or forged steel rolls, capable of withstanding high bending stresses, are used in blooming and slabbing mills, in the breakdown stands of section mills, and in cold-rolling mills. Forged rolls are somewhat stronger than the cast type but are from 1.5 to 2 times more expensive. Alloy steel rolls are used in sheet and plate mills; such rolls are chiefly of chrome-nickel or chrome-molybdenum steel.

The necks of rolls run in *sleeve (plain) bearings* or in *roller bearings* mounted in the chocks of the roll stand. Sleeve bearings are most frequently made of laminated fabric (textolite), plasticised wood or other materials. Water is used as the lubricant for laminated fabric bearings.

**Rolling mill classification based on roll stand design.** According to the number and arrangement of the rolls, mill stands can be classified into five groups:

- (1) two-high stands;
- (2) three-high stands;
- (3) four-high stands;

- (4) multiple-roll stands;
- (5) universal roll stands.

A *two-high nonreversing stand* has two rolls with a constant direction of rotation around horizontal axes (Fig. 95a). Such stands are widely used in mills through which the bar passes only once (open-train mills with finishing stands, mills with a consecutive arrangement of stands and others) and in open-train plate mills.

In the *two-high reversing stand* the rolls rotate first in one direction and then in the other so that the rolled metal may pass back and forth through the rolls several times. These stands are employed in blooming and slabbing mills, and as roughing stands of plate, universal, rail and structural, and other mills.

A *three-high stand* has three rolls with a constant direction of rotation, arranged in a single vertical plane (Fig. 95b). Tilting tables are provided on one or both sides of the stand to raise the bar and to enter it between the top and middle rolls. Stands of this type find extensive application in open-train section mills.

*Four-high stands* (Fig. 95c) have four rolls, one above another (in a vertical plane). The two rolls smaller in diameter are the working rolls while the larger are back-up rolls whose purpose is to support the working rolls and reduce their spring.

Four-high stands are used in reversing mills for the hot rolling of armour and other plate, as well as for the cold rolling of sheet steel in the widely used continuous mills for the hot and cold rolling of sheet and strip.

*Multiple-roll six-, twelve- and twenty-high stands* (Fig. 95d) have been widely employed in recent years. The very small diameter of the working rolls (10 to 30 mm) and the rigidity of the roll stand enable the thinnest strip to be cold rolled in coils. The working rolls are idle running (their small diameter makes a drive practically impossible); they are supported by a row of driving rolls which, in turn, are suppor-

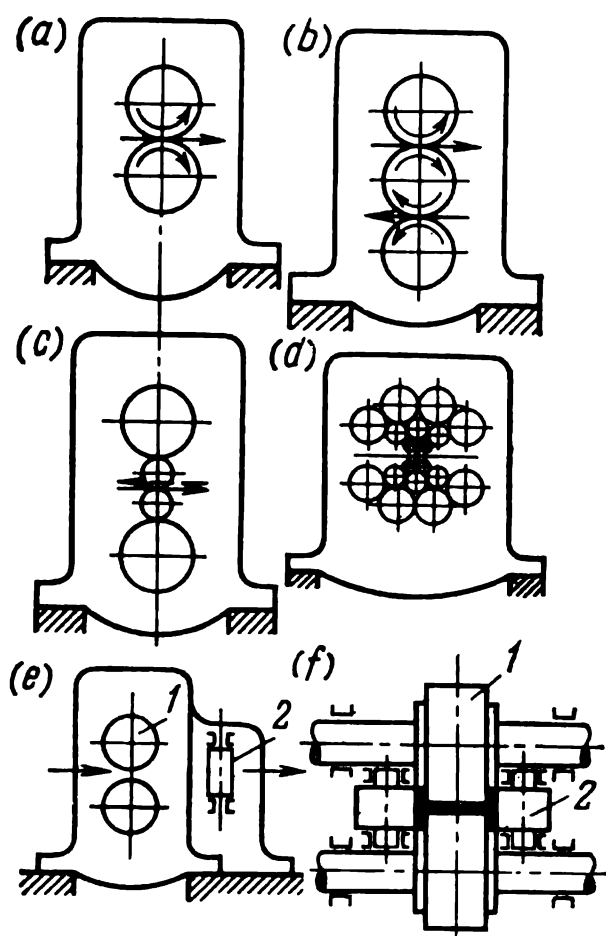


Fig. 95. Types of roll arrangement in rolling mill stands:

1—horizontal rolls; 2—vertical rolls

ted by a row of back-up rolls. This arrangement ensures exceptional rigidity of the whole roll system and an almost complete absence of working roll deflections.

*Universal roll stands* (Fig. 95e) are used for rolling wide strip, sheets, plates and slabs. Here the metal is reduced by both horizontal and vertical rolls; the latter roll the edges of the bar even and smooth. Vertical rolls are mounted either on one side (front or back) or on both sides of the horizontal roll stand. The horizontal rolls may be of either two-, three- or four-high arrangement.

Universal roll stands are also used to produce wide-flanged (H) beams with a depth up to 1000 mm in universal beam mills (Fig. 95f). The vertical rolls of universal structural mills are idle running and are arranged between the bearing chocks of the horizontal rolls in the same vertical plane.

**Mill drive motors and associated drive mechanisms.** At the present time mills are driven by d-c or a-c electric motors (induction or synchronous type). A heavy flywheel is mounted on the shaft of the mill motor (see Fig. 92, item 6) to smooth down peak loads in rolling and to enable a motor of lower horsepower to be employed.

Since the running speed of high-speed motors does not usually correspond to that of the rolls in the stands, a reducing gear is installed between the motor and the stand.

It is necessary to transmit the torque of the motor simultaneously to several rolls mounted in one stand. This is done by means of a pinion stand.

The shafts of the motor, reducing gear, pinion stand and the rolls are linked together by spindles and couplings to transmit the motor torque.

All auxiliary and handling machines and mechanisms of a rolling mill are also driven by electric motors at the present time.

*Rolling mill classification based on the products rolled.* According to their purpose, rolling mills may be classified as blooming, slabbing, billet, rail and structural, section, strip and skelp, wire rod, sheet and plate, universal, tube, and tyre and wheel mills.

*Blooming and slabbing mills* are heavy primary mills with rolls from 800 to 1400 mm in diameter. They are designed for rolling ingots from 2 to 25 tons in weight and heavier into blooms, slabs and shaped billets of large size. These serve as the initial material for heavy section, plate and billet mills.

*Billet mills*, with a roll diameter of from 450 to 850 mm, are designed for the further rolling of blooms into billets of smaller size (from 50×50 to 150×150 mm in cross section). These billets are used in the rolling of sections and wire rod. The most up-to-date billet mills are of continuous design and are installed directly after the blooming mills in the billet shop of the rolling department.

*Rail and structural mills* have rolls from 750 to 800 mm in diameter and are used mainly to produce railroad rails, beams, channels and other heavy structural shapes.

*Section mills* are further classified, according to the size of section they accommodate, into: heavy-section mills with rolls from 500 to 750 mm in diameter for heavy shapes and sections; medium-section mills with rolls from 350 to 500 mm in diameter for shapes and sections of medium size, and light-section, or bar, mills with rolls from 250 to 350 mm in diameter for light bar stock.

The simplest section mill is the open-train type in which all the roll stands are arranged side by side in a single line.

One vital disadvantage of these mills is that the rolls of all the stands run at the same speed. Therefore, it is impossible in these mills to roll the bar at a speed that increases with the length of the bar as it is successively reduced (and elongated).

Since the output of rolling mills can be increased with the differentiation of the production process the roll stands were subsequently designed in several lines.

The most effective section mills are the so-called *continuous mills*. The roll stands of such mills are arranged in tandem (one after another). The distance between the stands is usually less than the length of the bar being rolled so that the latter may be simultaneously reduced in all or several stands. The rolling speed increases in the successive stands as the cross section of the bar is reduced.

High output and complete elimination of manual labour are achieved in continuous mills. The high degree of automaticity in these mills enables much higher rolling speeds to be used (up to 30-40 m per sec or even more) than in open-train manual-control mills.

Mills with a zigzag arrangement of the stands, often called cross-country mills, are employed to produce medium- and heavy-section steel stock. Here, each stand is also used for a single pass but the bar is not rolled simultaneously in several stands. Between passes the bar is on the roll table between the consecutive stands. To keep the length of the building within feasible limits, the stands are arranged in three parallel lines and the bar being rolled is transferred from line to line by skew roll tables or transfers.

A modification of the cross-country mill is the staggered rolling train in which the finishing stands are arranged in staggered rows.

Semicontinuous mills are in use for the rolling of bar stock. They have a continuous roughing train and an open-type finishing train consisting of several stands.

Up-to-date continuous bar mills are equipped with an individual drive for each roll stand powered by motors with a wide range of speed adjustment. This enables the rolling speed to be set precisely in each stand. Vertical roll stands are also included in such mills;



this excludes the need for turning the bar between passes and the use of relatively unreliable twisting guides.

*Rod mills*, having rolls about 250 mm in diameter, are used to produce wire rod from 6 to 10 mm in size.

Modern rod mills are of continuous and semicontinuous design.

*Sheet and plate mills, for the hot rolling* of steel into sheets and plates from 1.25 to 60 mm and more in thickness, have rolls with barrel lengths ranging from 800 to 5000 mm.

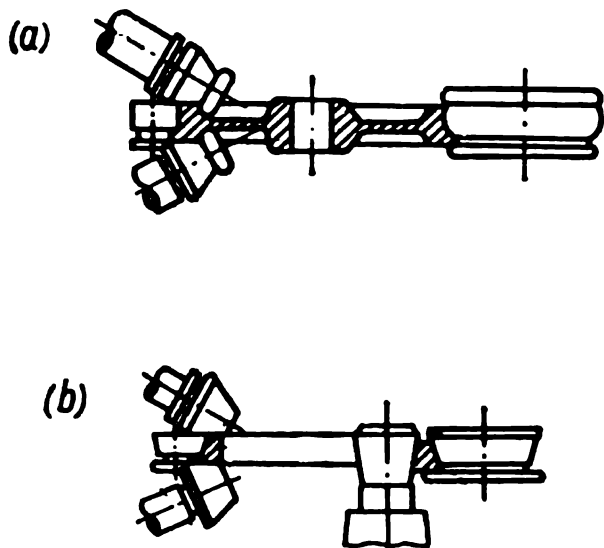


Fig. 96. Principle of the wheel (a) and tyre (b) rolling mills

Plates from 1500 to 2500 mm wide are rolled in two-stand mills. The stands are in tandem arrangement; one is the roughing and the other, the finishing stand.

At the present time, continuous and semicontinuous mills are being more and more extensively used in the production of sheet and plate.

*Cold reduction mills* are employed for the cold rolling of sheet steel from 0.05 to 4 mm in thickness; they have rolls with barrel lengths from 300 to 2800 mm.

Four-, twelve- and twenty-high mills find wide application in the cold rolling of thin strip of various grades of steel and nonferrous metals.

*Universal mills* produce the so-called universal mill plate which is from 200 to 1500 mm in width. The side edges of this plate are worked by the vertical rolls.

*Seamless tube mills*, as their name implies, produce seamless steel tubing. Seamless tubes are rolled in two operations: (1) piercing an ingot or billet to obtain a thick-walled shell and (2) reducing the shell by rolling into a tube of the required size.

In the production of welded pipe (with a longitudinal butt weld) wide use at present is made of continuous mills in which steel tubing is made by an electric welding process.

*Tyre and wheel mills* are used to manufacture railroad wheels and tyres by a rolling process. The principle of wheel rolling is illustrated in Fig. 96a, and that of tyre rolling in Fig. 96b. Both of these methods are combined mechanical working processes, involving both forging and rolling. In tyre production, first a tyre billet is obtained by forging and then it is rolled into the finished tyre. Wheel billets are die-forged in a press and further processed by rolling to obtain the finished wheel.

**Auxiliary and handling equipment of rolling departments.** Metal rolled in the mill undergoes various finishing operations, such as cropping the ends of the bar, cutting into the required lengths, straightening, coiling wire rod and strip steel, etc.

All stock finishing operations are mechanised and are performed by auxiliary machines and mechanisms driven by electric motors.

Auxiliary equipment for rolling mills serves many purposes and is available in a great variety of types and designs which include stationary and flying shears, hot saws, straighteners, strip coilers, wire coilers, cooling beds, etc.

In the rolling process, the metal being rolled must be transported along the shop bays and in a crosswise direction from one mechanism to another, handled at the various machines, raised and lowered, and certain other operations must be performed. All this can be done at an ample speed if high-production handling equipment is available. In some cases, the rolled stock is subjected to processing operations while it is being transported from one place in the shop to another. For this purpose, use is made of various types of handling equipment such as apron, flight and roller conveyers; roll tables; manipulators; pushers; billet ejectors; and slab and billet pilers.

## Chapter 19

### METAL ROLLING PRACTICE

#### 19-1. Metal Rolling Procedures

**Types and sizes of rolled products.** Rolled products can be divided into four main groups: (1) steel shapes or sections, (2) plate and sheet steel, (3) special-purpose rolled shapes and (4) tubing.

*Steel sections* are widely used in almost all branches of industry and can be further classified into general-purpose and special sections. The first group consists of rounds, squares, flats, angles, strip, wire rod, channels, I-beams (including light-weight and wide-flange H beams), and others (Fig. 97). In the second group are rails and special shapes employed in construction, engineering and other fields of industry.

*Plate and sheet steels* are classified according to their thickness. All flat stock over 4 mm thick is, as a rule, called plate while comparatively wide stock, up to and including 4 mm in thickness, is called sheet steel.

*Special-purpose rolled shapes* include tyres, one-piece rolled wheels and die-rolled products in which the cross section of the bar varies periodically along its length. Die-rolled products (for instance, reinforcement bars) are used in the building industry for making reinforced concrete.

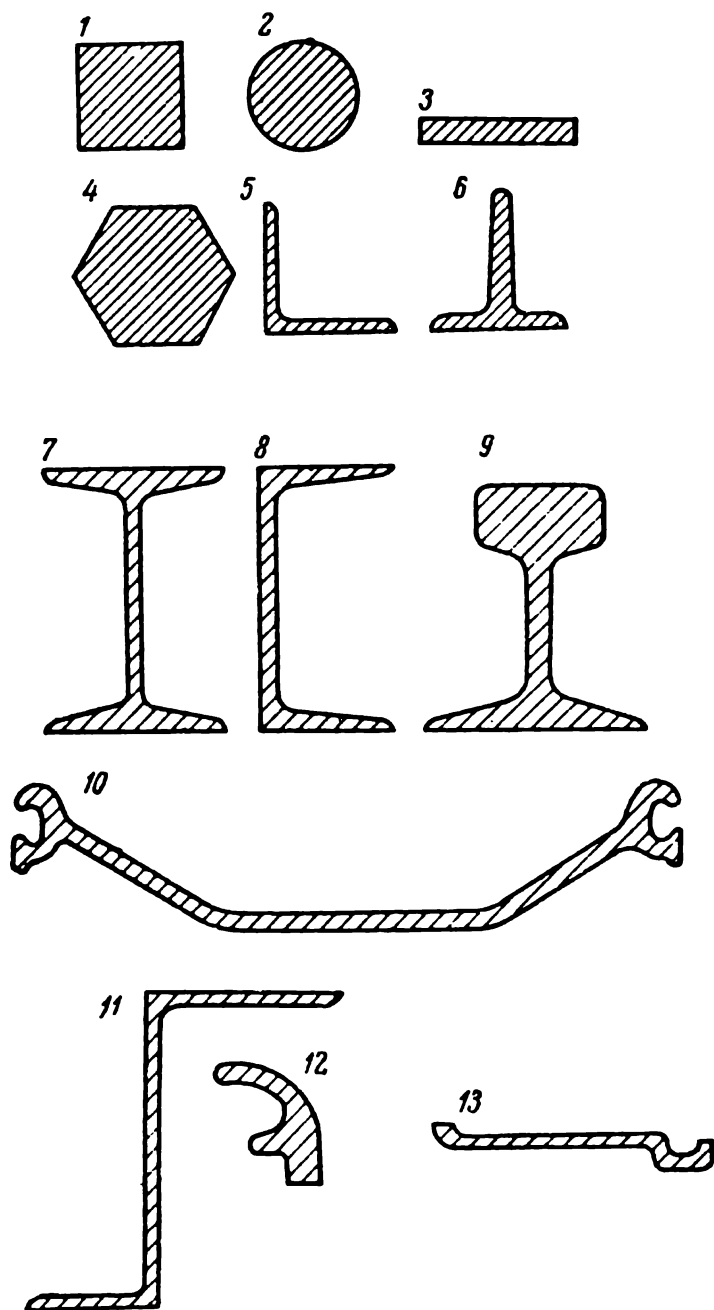


Fig. 97. Steel sections:

1—square; 2—round; 3—flat; 4—hexagon; 5—angle; 6—tee; 7—I-beam; 8—channel; 9—rail; 10—sheet piling; 11—zee; 12—automobile tyre-rim locking ring; 13—tyre rim

Still another type of special-purpose product is the cold-roll-formed shape. The use of such shapes enables processing to be considerably simplified in the manufacture of various parts.

A great variety of cold-roll-formed shapes are made of sheet, strip or plate from 0.2 to 20 mm thick; they find especially wide application in construction (various structural elements, window sashes, etc.).

*Steel tubing* may be of three main types: (1) seamless, (2) furnace-welded and (3) electric-welded.

The production of welded and cold rolled tubing has grown rapidly in the last years.

**Processing operations in rolling practice.** The main operations in metal rolling are: preparing the initial material, heating the material before rolling, rolling and finishing.

*Preparation of the initial material* for rolling consists in the removal of various surface defects (also called

conditioning). This is a very important operation, especially in the rolling of high-quality carbon and alloy steels, as it ensures a high yield of proper quality with minimum rejects.

Strict observance of the stipulated conditions for *heating the metal* before rolling, established and checked by practice and differing for

each grade of steel or group of grades; proper selection of the temperatures for the beginning and end of the rolling process; and an optimum draughting schedule are of vital importance and directly affect, not only the quality of the finished product, but the mill output as well. Nonobservance of the prescribed heating conditions may be the reason why the finished product has unsatisfactory mechanical properties.

Process control procedures during *rolling* include the checking of the initial and final rolling temperatures and whether the specified draughting schedule is being observed. The setting of the rolls is checked constantly during rolling by inspection of the shape and size of the bar delivered by the rolls. The condition of the roll passes, as well as the setting and surface condition of the guiding fixtures must also be inspected frequently. Recently, attention has been paid to a determination of the rolling forces (roll load) so that the available power of the mill can be more effectively and fully utilised.

Surface quality inspection is done by regularly taking samples of the rolled product and revealing all the surface defects. In some cases, the samples are etched and inspected again. More thorough surface inspection is conducted before conditioning the rolled stock.

Process control in the *finishing operations* following the rolling process consists in inspection after cutting up the bar into lengths, after straightening, after removing surface defects, etc.

The rolling process is controlled at all of its stages.

Final inspection concerns the finished product. Its purpose is to determine whether the quality of the rolled product complies with the stipulated specifications and other requirements. Sampling procedures are employed in final inspection. If the sample taken does not pass the requirements, a new sample in which the number of test pieces is doubled may be taken for a second inspection.

## 19-2. The Rolling of Blooms, Slabs, Billets and Heavy Sections

**The rolling of blooms and slabs.** A characteristic feature of modern rolling practice is the provision of blooming or slabbing mills. These primary (breaking-down) mills are installed between the steelmaking department and the rolling mills that produce the finished product.

Blooming and slabbing mills roll square or rectangular ingots into heavy billets called *blooms* and *slabs*.

The weight and shape of the ingots depend upon the roll diameter in the mill, the available power of the mill drive, the grade of steel and the type of semifinished product.

The weight of the ingot should be such as to ensure the maximum output of the mill, proper quality of the blooms or slabs, and mini-

mum rejects and discards. Considerably heavier ingots are used to produce slabs, for example, than for blooms.

To ensure a high tonnage in the department, blooming mills should produce heavy blooms with a cross section of at least  $200 \times 200$  mm and up to  $350 \times 350$  mm.

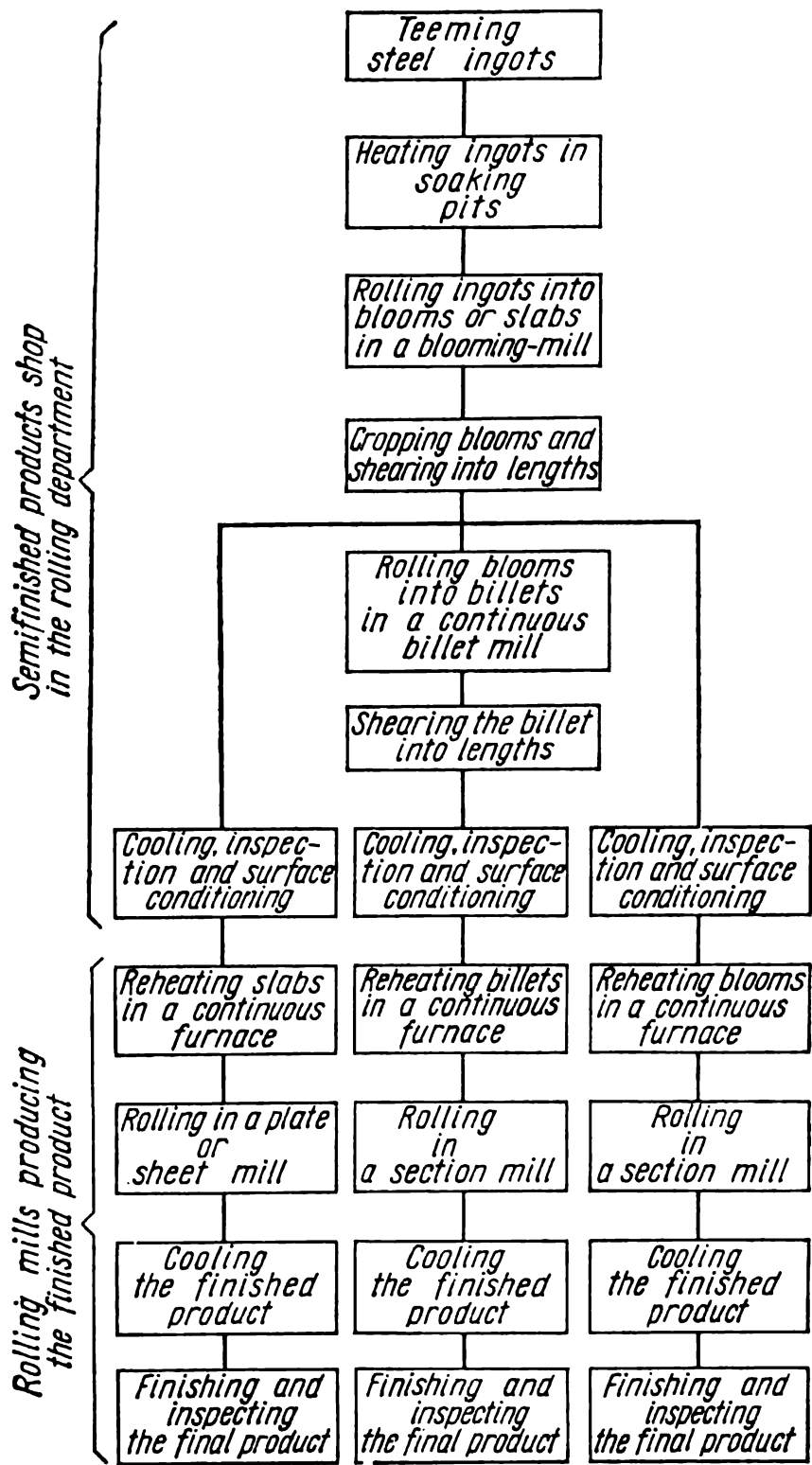


Fig. 98. Flow diagram of modern rolled stock production from the ingot to the finished product

A flow diagram of bloom production, beginning with the ingot and up to the finished product, is shown in Fig. 98.

Soaking pits are used exclusively for preheating the ingots before rolling them into blooms or slabs. About 90 per cent of the ingots charged into the soaking pits of blooming and slabbing mills are transferred hot directly from the steel-making department, i. e., immediately after they are stripped from the ingot moulds, at an average temperature of  $800^{\circ}$  to  $850^{\circ}$  C.

As a rule, the soaking pits are located in a separate building adjacent to the blooming or slabbing mill building.

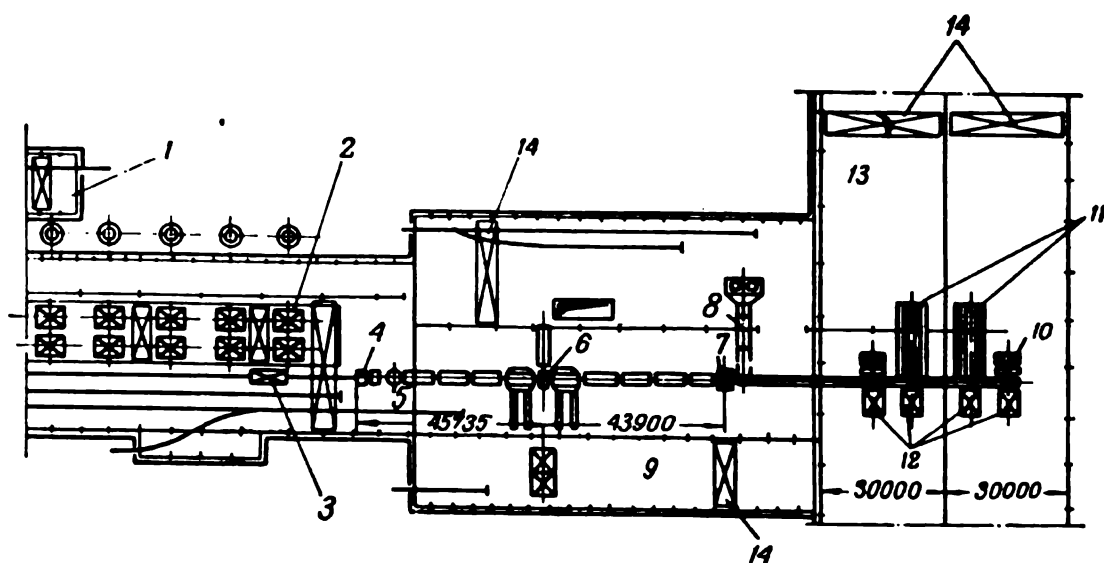


Fig. 99. Layout of a modern 1150-mm single-stand blooming mill:

1—room for delivering coke breeze and for clinder disposal; 2—soaking pits; 3—ingot cart; 4—receiving roll table with a stationary tilter; 5—turntable; 6—roll stand; 7—shear; 8—scrap conveyor; 9—motor room; 10—piling tables; 11—cooling beds; 12—pushers; 13—bloom and slab storeroom; 14—cranes

Fig. 99 illustrates the equipment layout of a modern 1150-mm single-stand two-high reversing blooming mill (1150 mm is the roll diameter). The soaking pit room is an extension of the main mill building and comprises two bays, one housing the pits themselves and the other containing equipment for gas and air supply and for pit control.

The output of blooming mills has increased considerably in recent years and they roll 2.5 or 3 million tons of ingots per year or even more in some cases. At such a rate of production the time required to roll one ingot is from 50 to 60 seconds.

Blooming mills are usually powered by d-c motors with a rating up to 7000 hp and with adjustable speed (0-50-120 rpm).

The blooming mill roll pass design illustrated in Fig. 100 has found widespread application in Soviet metallurgical plants.

Data collected in leading plants in the U.S.S.R. indicate that the reduction of the ingot in one pass, when blooms are produced

of carbon and low-alloy steels, ranges from 65 to 75 mm on the average. The maximum permissible reduction may be as high as 80 or 120 mm. A schedule based upon such draughting practice provides for rolling a carbon steel ingot weighing 7 tons into a bloom with a cross section of  $250 \times 250$  mm in 15 passes and an alloy steel ingot weighing 6.3 tons into a bloom of the same size in 13 passes.

The yield of proper quality blooms from rimmed steel ingots is from 91 to 92.5 per cent; the yield from rail steel ingots is from 80 to 82 per cent.

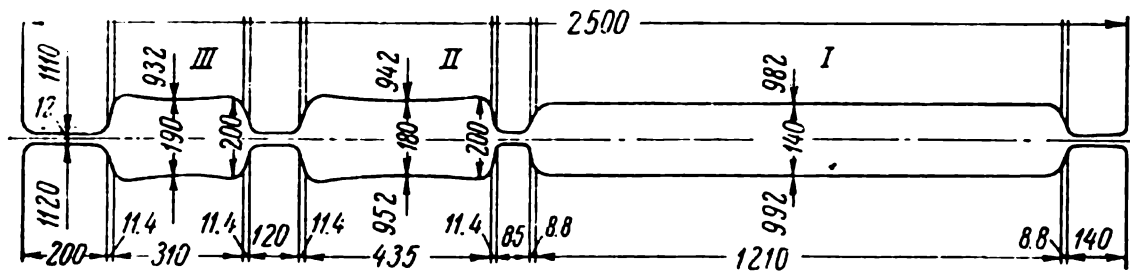


Fig. 100. Roll pass design for the 1150-mm blooming mill:

I, II and III—numbers of the consecutive passes

Blooms are further processed either in rail and structural mills to produce heavy shapes or in billet mills to produce billets which are subsequently rolled into bar, rod and certain medium and light sections.

Billet mills are usually located in the immediate vicinity of blooming mills and are included in the primary or roughing shop of the rolling department. A mill layout of this type enables heavy ingots to be rolled into light billets without reheating. This is obviously good practice insofar as production costs are concerned.

**The rolling of steel shapes in rail and structural mills.** The finished products of rail and structural mills are railroad and streetcar rails, I-beams, channels, sheet piling, and angles, rounds and squares of heavy cross section.

Rail and structural mills produce up to 10 or 12 per cent of the total amount of stock rolled in the U.S.S.R. in the form of standard and heavy rails, structural I-beams, sheet piling and other heavy shapes.

Standard structural beams commonly rolled in rail and structural mills have a depth range from 180 to 600 mm. During the last few years, the same mills have begun to produce beams with a depth from 650 to 750 mm, light-weight thin-webbed beams with a depth up to 300 mm and with less taper on the flanges, and wide-flange beams (H-beams) with a depth up to 300 mm and more. The latter are employed in construction as columns and other structural elements.

Sheet piling may be of the flat, arched and zee types depending upon its purpose in hydraulic structures (dams, antifiltration curtains, etc.).

Modern rail and structural mills are usually arranged in two or more lines and comprise several roll stands (Fig. 101).

The first line of this mill is the two-high reversing breakdown stand 1. In design, this stand resembles a blooming mill; its rolls are commonly from 900 to 950 mm in diameter and have a barrel length of 2300 mm. Devices for transferring and turning the bar,

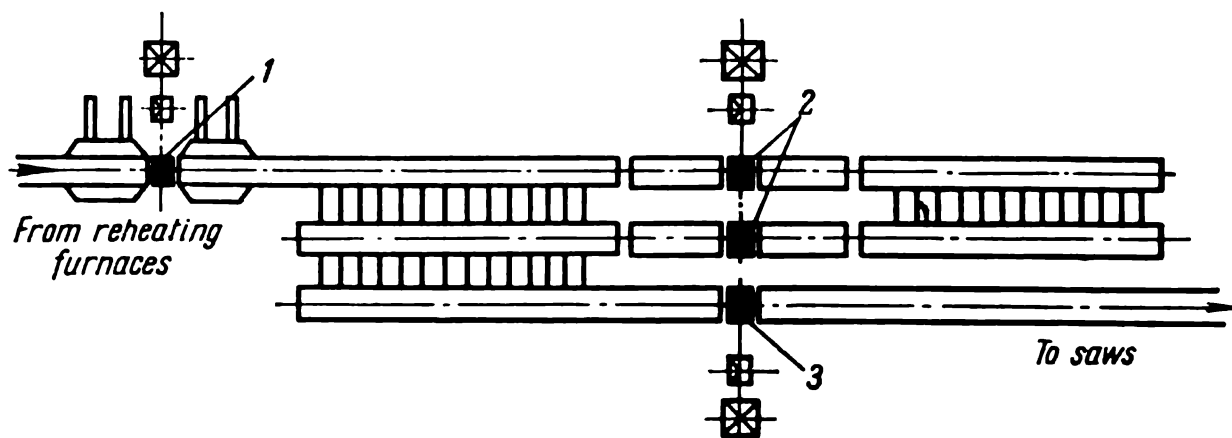


Fig. 101. Rail and structural mill with the stands arranged in two lines

manipulators and tilters, are provided on both sides of the stand to facilitate handling in the rolling process. The drive of the first line (950-mm two-high breakdown stand) is powered by a 5000-hp reversible motor.

The second line of rail and structural mills usually consists of an open train of two 850-mm three-high roughing stands 2 powered by a single 6000-hp reversible motor.

As a rule, the two-high finishing stand 3 has rolls of the same diameter as the roughing stands. This open-train stand is equipped with all the necessary guiding fixtures and, in many cases, its rolls run in antifriction bearings which ensure high rolling accuracy. A 2500-hp d-c motor drives the finishing stand.

Light-weight structural beams are rolled in rail and structural mills equipped with an interchangeable universal roll stand which may be installed in place of the two-high finishing stand.

**Roll pass design for rail and structural mills.** Various pass sequences may be employed in these mills depending upon the size and shape of the beam to be produced. The more typical beam rolling sequences are shown in Fig. 102.

Sequence 1 consists only of closed beam passes. The first, a web-cutting pass with sharp cutting elements, performs the initial deformation. Here, the initial billet, usually of rectangular cross se-



ction, is cut by the sharp web-forming elements, above and below, to form the initial rough beam section. In the subsequent passes, the cuts are gradually deepened and widened to form the web and to reduce the flange thickness. This sequence is common practice for the rolling of beams in sizes from No. 10 to No. 30 (depths from 100 to 300 mm).

Medium and heavy beams, in sizes from No. 30 to No. 60, are rolled by sequence *II* in which the beam is formed in open beam passes.

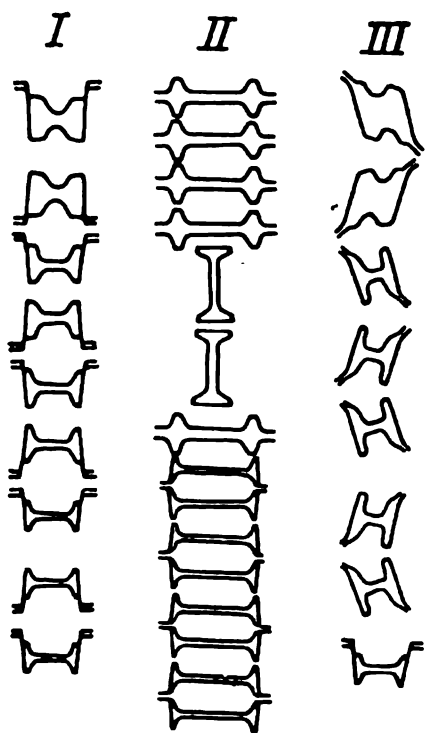


Fig. 102. Typical pass sequences for the rolling of beams

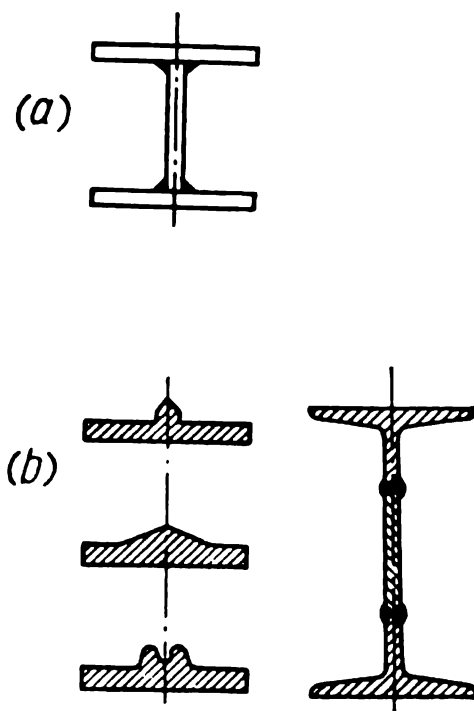


Fig. 103. Welded I-beams: (a) welded of flat elements; (b) with a flat web and two specially shaped flange members

The pass grooves are cut into the blooming mill rolls and in the rolls of the two-high reversing roughing stand of a rail and structural mill. Several passes of the bar are made in each groove by changing the distance between the rolls for each pass. Overfills, formed on the sides of the bar after several passes, are eliminated by turning the bar  $90^\circ$  and passing it through the plain part of the barrel. Deformation in open passes continues until the web thickness is reduced to a value from three to five times the final thickness. The ratio of the web and flange thicknesses in the rough section is maintained the same as in the finished beam. Further rolling is performed in closed (tongue and groove) beam passes.

Sequence *III*, which uses closed beam passes inclined to the horizontal (diagonal passes), has found much application recently.

This method enables beams with parallel flanges or with a very small flange taper to be effectively rolled.

Diagonal pass design, which permits heavy draughting to be applied and the beam to be rolled in less passes, is used in the rolling of light-weight and thin-webbed beams.

I-beams may also be manufactured by welding either three flat elements (Fig. 103a) or a flat web to two flange members (Fig. 103b).

Channels are produced by roll pass sequences such as shown in Fig. 104.

After rolling, rails, beams and other shapes undergo finishing operations, including cutting, cooling, straightening, etc., as well as testing and final inspection.

The consumption of blooms to produce one ton of beams may be taken as 1.06 tons.

### 19-3. The Rolling of Section Steel

Rounds, squares, flats and angles; light beams and channels; and wire rod are produced in section mills.

The billets for section steel are pre-heated in recuperative continuous furnaces of various designs.

The yield of the finished product in rolling billets in up-to-date section mills averages from 91 to 96 per cent; a higher yield is achieved in bar, strip and rod mills. The steel consumption in billets to produce one ton of rolled stock varies from 1.099 to 1.041 tons.

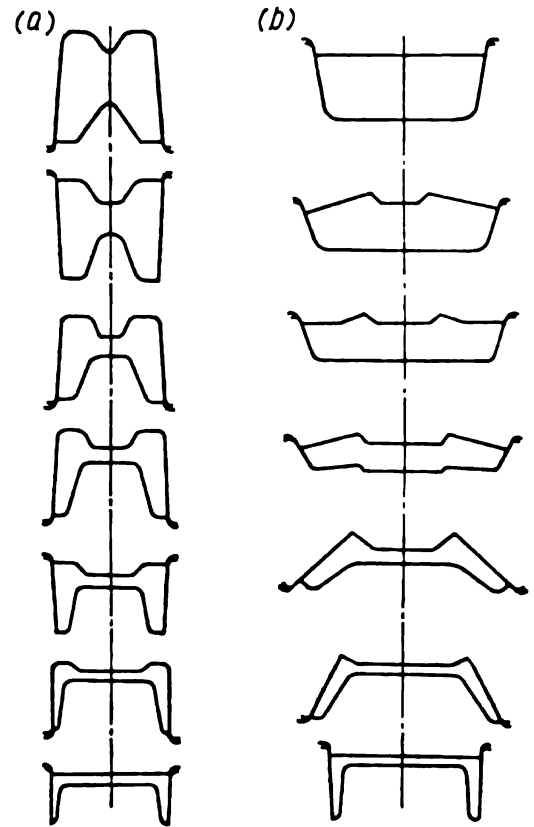


Fig. 104. Pass sequences for the rolling of channels:

(a) beam rolling method; (b) butterfly method

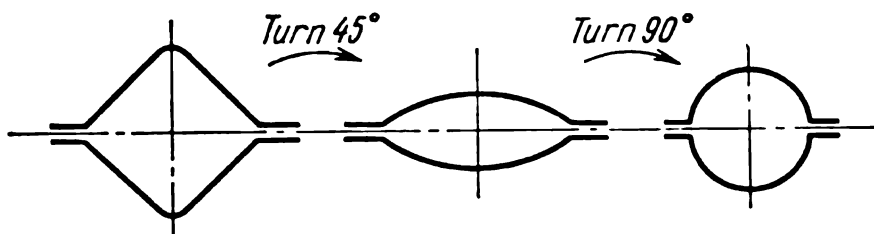


Fig. 105. Pass sequence for the rolling of rounds

**The rolling of rounds and die-rolled reinforcement bars.** Several different rolling sequences have been developed for producing *rounds* and *wire rod*. The most widely employed sequence, especially for rounds from 5 to 20 mm in diameter, is one that consists of a round finish-

ing pass, an oval leader pass and a square strand pass (the last pass but two) (Fig. 105). Various systems of breakdown passes—diamond-diamond, diamond-square or oval-square sequences—may be used for roughing.

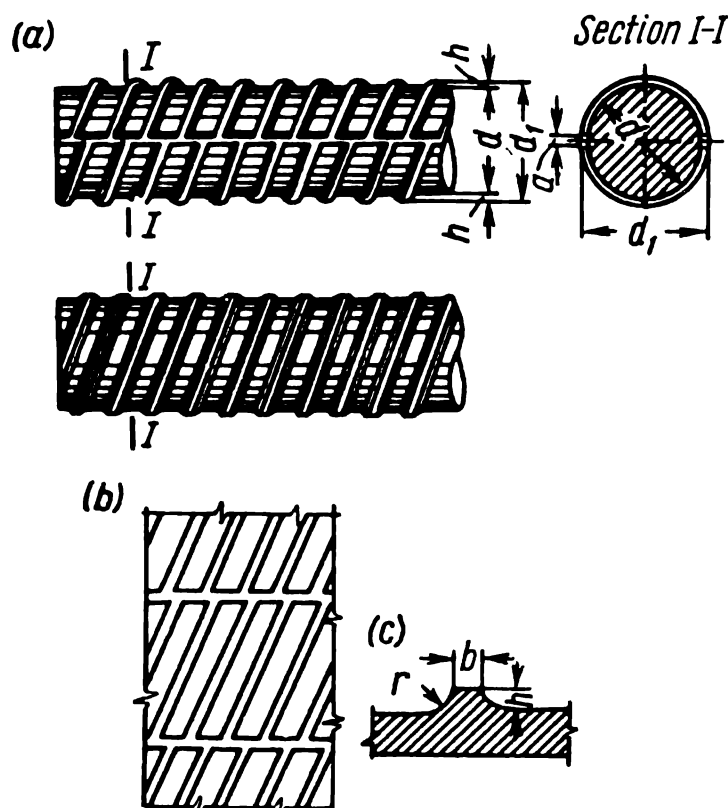


Fig. 106. Modern deformed (die-rolled) reinforcement bar:

(a) deformed bar; (b) developed view of the bar surface; (c) cross section of the helical ridge

Wire rod is produced with the application of the highest feasible reduction and at maximum speed; this is achieved by using the highly effective oval-square pass sequence.

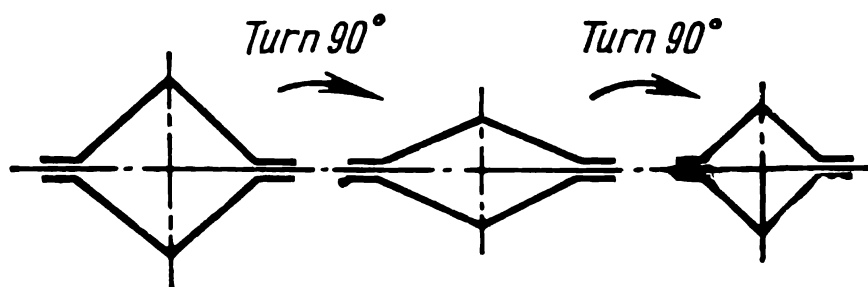


Fig. 107. Pass sequence for the rolling of squares

Section mills are also used in the die (periodic) rolling of *reinforcement bars* employed in reinforced concrete.

Die-rolled reinforcement bars are round bars with two longitudinal fins and ridges in the form of a three-start helix (Fig. 106).

The finishing pass grooves for reinforcement bars are turned in the rolls to the size of the minor diameter of the cross section while the helical impressions on the surface of the groove are obtained by milling. An oval bar is entered into this round pass with helical impressions.

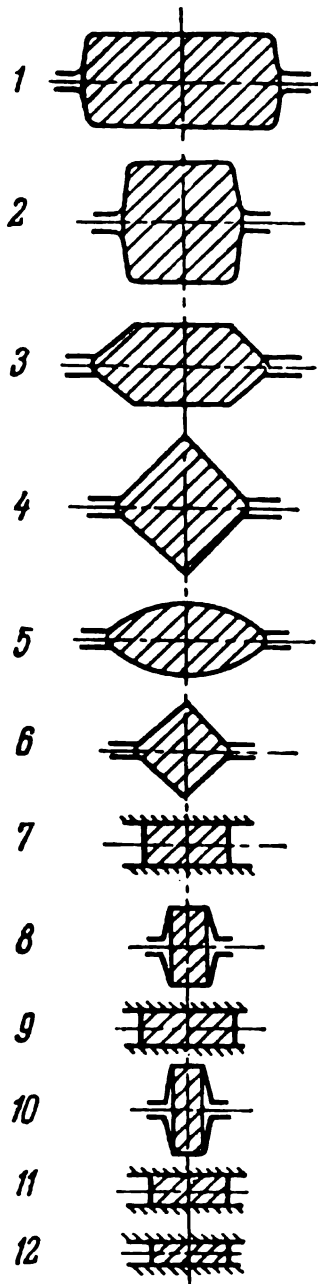


Fig. 108. Pass sequence  
for the rolling of flats:  
1 to 12—pass numbers

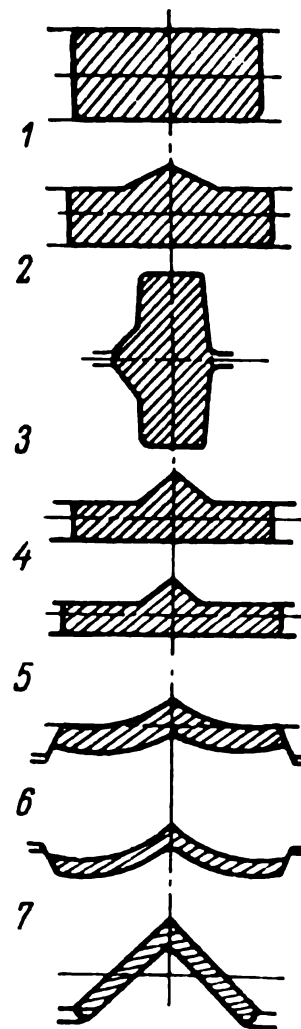


Fig. 109. Pass sequence  
for the rolling of angles:  
1 to 7—pass numbers

Thus, die rolling of reinforcement bars differs from the rolling of plain rounds only in the grooves of the finishing pass and in the somewhat larger oval leader and square strand passes. The latter are increased in size so that the extra metal will be forced into the milled impressions of the finishing pass.

Die-rolled reinforcement bars (commonly called deformed bars) in sizes from No. 10 to No. 90 (U.S.S.R. Std) find use in construction.

At the present time, bars of small diameter (6, 7, 8 and 9 mm) rolled from low-alloy steels are required for construction purposes. Calculations show that a metal savings of 20 per cent can be achieved without detriment to the strength of the structure if reinforcement bars of low-alloy steels are employed.

**The rolling of squares.** The most widely used sequence for rolling squares is the one shown in Fig. 107. It consists of a square strand, diamond leader and square finishing passes. Diamond-diamond and diamond-square breakdown sequences are regular practice for the roughing stands while an oval-square roughing sequence is used for squares of the smaller sizes.

**The rollings of flats and strip.** In modern section mills in which only one pass is accomplished in each stand, strip and flats are rolled in a sequence where plain-barrelled rolls alternate with edging passes. The number of edging passes and their order in the sequence depend upon the definite conditions involved in each case (Fig. 108).

The most efficient method of rolling flats and strip is that performed in specialised continuous mills with alternate horizontal and vertical roll stands.

**The rolling of angles.** The pass sequence illustrated in Fig. 109 has become widespread practice in the rolling of steel angles. The bar is rolled without spread restriction in the roughing passes and therefore the roughing rolls can be used for all sizes of angles.

#### 19-4. The Rolling of Flat Products

Plate and sheet mills produce: (1) *steel plate* of the bridge, tank, ship-building, armour, boiler and firebox types, and (2) *sheet-steel*—structural, roofing, tin and black plate, electrical, transformer and dynamo types.

**The rolling of steel plate.** Plate is rolled from slabs with a thickness from 65 to 300 mm, width from 600 to 1600 mm, length from 1000 to 2000 mm and weighing up to 2.2 tons.

Slabs are preheated in continuous furnaces.

The process of rolling plate comprises two stages: (1) rolling the slab broadside on until the required width is obtained; for this the slab is turned 90° in a horizontal plane, after one or two lengthwise passes through the rolls, and rolled crosswise (called broadside rolling or broad-siding) and (2) rolling the slab lengthwise again, after the required width has been obtained and the slab is turned back 90°, to the required thickness.

The draught per pass is determined on the basis of the bite conditions, strength of the rolls and the available power of the stand drive.

Plates are rolled in single-stand three-high, double-stand and semicontinuous mills.

**The rolling of sheet steel.** In most cases, sheet steel is produced by a combination of hot and cold rolling methods.

*Hot rolling* is done in semicontinuous and continuous mills. In these mills, slabs are rolled into sheets and plates from 600 to 2200 mm wide and wider, and from 1.25 to 12 mm thick.

Continuous mill equipment for the finishing of hot-rolled sheet steel includes a normalising furnace, pickling unit, drier, temper mills, straighteners and shears.

The finished product is in the form of cut sheets or coils.

*Cold rolling*, or cold reduction, the final operation in the production of thin sheet steel, is done by one of two methods. The first method, in which separate sheets are rolled, is used, at present, only in old metallurgical plants. The second method, rolling in coils, is up-to-date practice.

After being hot-rolled the coils of sheet steel are delivered to the pickling shop. The pickling line comprises a decoiler, shear for cropping the front end of the strip, welder or stitcher for joining successive coils, pickling and rinsing baths, hot-air drier, shear for cutting out the stitched seams and the coiler.

When pickling and other preparatory operations have been completed, the coils are delivered to the cold-reduction mills; lubricant is applied to the rolls and strip in rolling and the rolls are cooled.

Work hardening of the metal in the course of cold rolling reaches a stage where further reduction becomes difficult. Work hardening is removed by process annealing, carried out, as a rule in cover-type furnaces with controlled atmospheres or in normalising furnaces.

Following annealing or normalising the strip is trimmed and delivered for temper, or skin, rolling which is cold rolling with a reduction within 0.5 to 1.5 per cent per pass.

Various types of mills are employed in the cold rolling of sheet steel in accordance with the volume of production, purpose of the product and other factors. Four-high sheet mills and, less frequently, three-high mills are used in old plants with a relatively low volume of production. With the development of the continuous hot-rolling method of producing sheet steel and strip in coils, modern plants came to be equipped with four-high reversing mills and three- and five-stand continuous mills for cold-rolling operations. Further rolling of the strip in these mills is also done in coils.

Three-stand continuous mills, as well as four-high reversing mills, are used to roll sheet steel of all kinds except tin plate.

Tin plate in coils weighing up to 15 tons is rolled in five-stand continuous mills from hot-rolled strip about 2 mm thick; rolling

speeds are as high as 30 m per second. The rolling operations are the same as for sheet steel in coils.

Tin plate may undergo tinning, galvanising (zinc coating), lacquering or other operations after rolling, depending upon its purpose, as well as to protect it against corrosion.

Tinning may be done by either of two methods—by dipping in molten tin (hot dipping) and by the electrolytic process. Electrolytic tinning is a continuous process which ensures a more uniform coating and substantially reduces tin consumption.

Metal consumption varies widely in the production of flat rolled stock depending upon its purpose and thickness. It ranges from 1.24 to 1.6 tons of steel in ingots per ton of plate or sheet steel.

### **19-5. Cold-roll Forming of Metal**

Light-weight thin-walled shapes of intricate cross section may be produced by the cold-roll forming of sheet, strip or coiled stock.

Cold-roll-formed shapes have many advantages over hot-rolled shapes. They can be made with a thickness as small as 0.5 mm while in hot rolling in modern mills shapes with wall thicknesses less than 3 mm are very difficult to obtain. They may be of a very complex form with an expedient section modulus, thereby enabling a more effective cross section to be designed for each application so as to reduce the weight of the structure and to save metal.

Such shapes are made in cold-roll-forming machines of either periodic-action or continuous types. The initial material for the first type is comparatively short sheet or strip stock; the second type accommodates coiled stock. The stock is fed longitudinally through successive pairs of driven rolls, each pair of rolls progressively forming the stock.

These machines are available in two designs: (1) with overhung roll spindles and (2) with rolls between housings supporting both ends of the roll shafts. The overhung spindle type is used only for forming light-gauge shapes.

The number of pairs of rolls (stands) in the machine depends on the shape being produced; the more intricate the shape, the more rolls are required for its forming. The peripheral speed of the rolls in various machines may vary from 20 to 120 m per minute, the lower speeds being for the heavier machines.

The automobile industry employs cold-roll-formed shapes of small sizes, for the most part, such as are used for windshield frames, radiator tubes, etc. They are formed of thin (0.2 to 1 mm) and narrow (16 to 80 mm) steel strip.

In construction these products find application in the form of channel and beam sections, stamped sheets and others.

## 19-6. Manufacture of Tubular Products

**Seamless tube rolling processes.** Seamless tubes are produced in pilger, plug and continuous tube mills.

The initial material for *pilger mill* tubes are round or polyhedral ingots from 250 to 600 mm in diameter and weighing from 0.6 to 3 tons. Hollow tube shells are obtained by piercing round ingots or polyhedral ingots with a large number of sides, in a roll piercing mill. Ingots with a small number of sides are usually prepierced in a press. The roll piercing mill, or cross-rolling mill, has two inclined work rolls rotating in the same direction (Fig. 110) and two idle guide rolls.

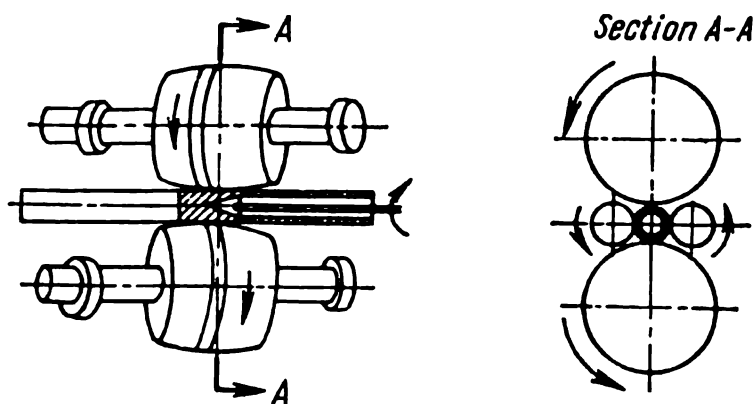


Fig. 110. Piercing the tube billet in a barrel-type roll piercing mill to obtain the shell

The work rolls of piercing mills are combinations of two or more truncated cones with intermediate cylindrical sections. The angle of inclination of the axes of the work rolls varies from  $4^{\circ}30'$  to  $6^{\circ}30''$ .

The billet entering the piercing mill makes a small angle with each of the work rolls. Because of the inclined position of the rolls in reference to the billet, the motions imparted to the latter are simultaneous rotation and axial advance.

In the rolling process a cavity is formed in the central zone of the ingot or billet where the metal becomes friable due to the cross rolling action. The required cylindrical form is imparted to this cavity by the mandrel over which the material is drawn. The mandrel is set between the rolls facing the direction of metal travel and its diameter should be equal to that of the pierced hole in the tube shell.

Further forming of the shell into a tube of the specified size is done in a pilger mill which has rolls with pass grooves of varying profile (Fig. 111). The rolls rotate in opposite directions at the same speed; the direction the shell is fed by the rolls is opposite to the direction it is fed between the rolls by the feeder. Upon rotation of the rolls, the dimensions of the pass change continuously and, therefore, the form of the pass is variable during one revolution of the rolls. At the maxi-



imum diameter of the pass groove the rolls form an idle pass (position 1) whose dimensions are larger than the diameter of the shell. At this moment, the shell together with the inserted mandrel is automatically advanced between the rolls by the amount of feed. Upon further rotation of the rolls the dimensions of the pass are gradually decreased due to its varying form and the shell is reduced (position 2), this re-

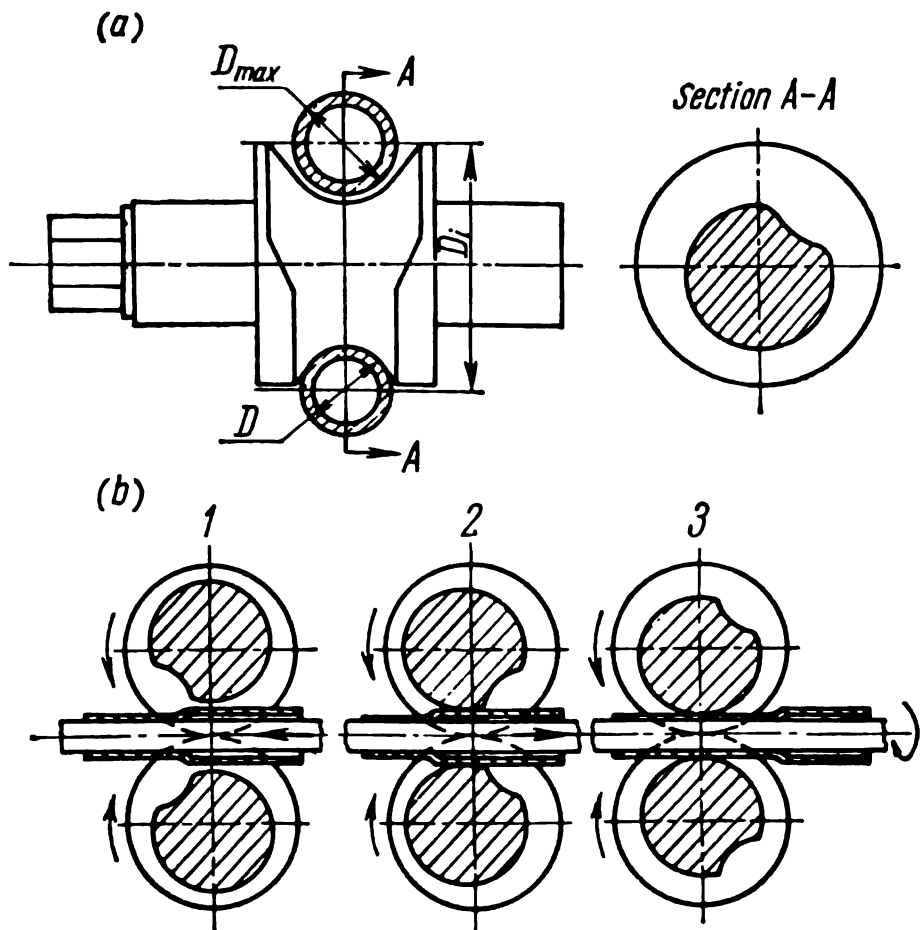


Fig. 111. The pilger rolling principle illustrated for one revolution of the rolls:  
(a) position of the shell in respect to a roll at the moment of feed into the pass and at the end of the rolling cycle; (b) rolling of an annular section of the tube

duction being increased with the decrease in the pass groove diameter. The rolls force the gripped annular section of the shell in the direction of roll rotation so that the shell and mandrel are shifted backwards (opposite to the direction they are fed by the feeder, position 3). When the rolls have rotated through  $360^\circ$ , they return the idle pass and the feeder again advances the shell between the rolls. The shell is turned  $90^\circ$  as it is being fed forward. After rolling the shell, the mandrel is removed from the tube; the next shell is rolled on a new mandrel while the preceding one is cooled, lubricated and prepared for further use.

In addition to mills with barrel-type rolls shown in Fig. 110 tube billets are pierced to obtain shells in  $60^\circ$  cone roll mills (Fig. 112a) and  $180^\circ$  disk piercing mills (Fig. 112b).

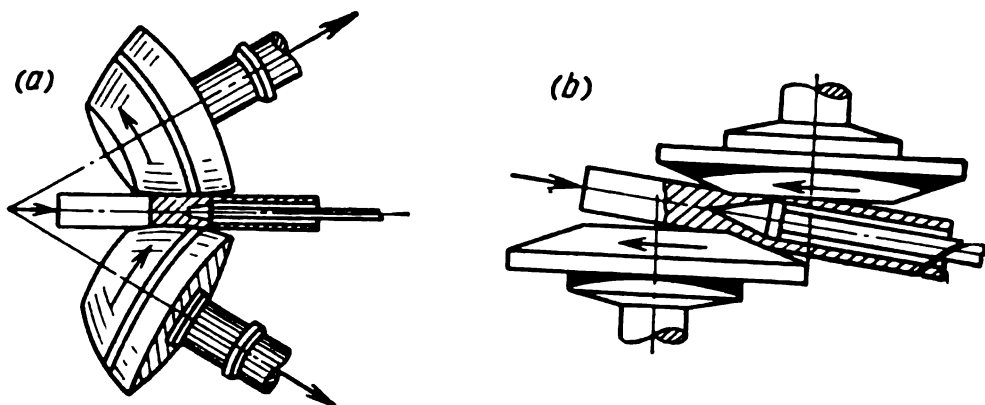


Fig. 112. Billet piercing mills:  
(a)  $60^\circ$  cone roll mill; (b)  $180^\circ$  disk mill

Another method of further processing the pierced shell is in a *plug mill* which is a two-high reversing stand with a series of round passes in the rolls. A short mandrel is held in the centre of the pass (Fig. 113) by a long bar. The clearance between the mandrel and the pass determines the wall thickness of the rolled tube.

Tubes are usually rolled in two passes in the plug mill regardless of the tube wall thickness. In the next operation the tube is rolled in a reeling machine which improves the finish of the inside and outside surfaces; eliminates irregularities, scratches and out-of-roundness (ovality), and decreases the differences in wall thickness. In reeling, the diameter of thin-walled tubes increases considerably while that of thick-walled tubes increases to a lesser extent.

The reeled tubes are then delivered to the sizing mill which rolls them to the specified diameter.

The shells for tubes produced in a *continuous seamless tube mill* (Fig. 114) are obtained in ordinary piercing mills.

Continuous tube mills are available in two principal designs: (1) with a common drive for all the roll stands and (2) with an individual drive for each roll stand.

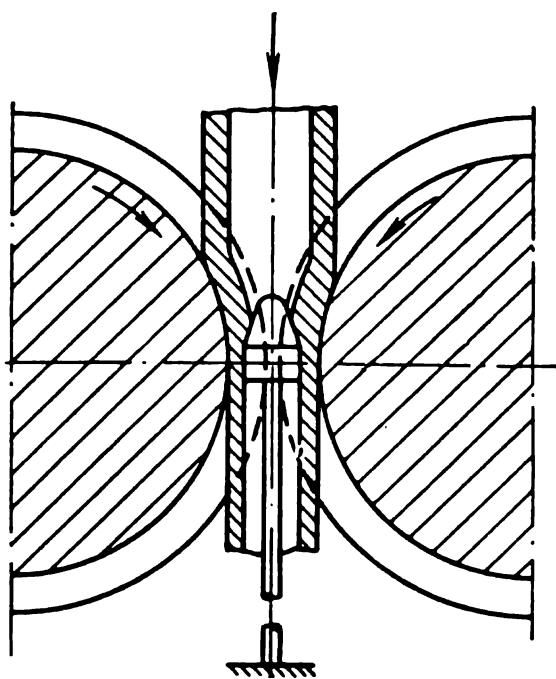


Fig. 113. Principle of tube rolling in a plug mill

After rolling in a continuous mill the tubes are either sized or reduced (to change their diameter). The development of the stretch reducing mill has enabled the size range of tubes rolled in continuous mills to be cut to a minimum, thereby substantially increasing the output of the continuous mills.

**The production of welded steel tubing.** Welded tubing is made of hot-rolled strip steel, known as *skelp*.

The production of welded tubing consists of two operations: (1) forming the skelp into the tube and (2) welding the joint of the tube.

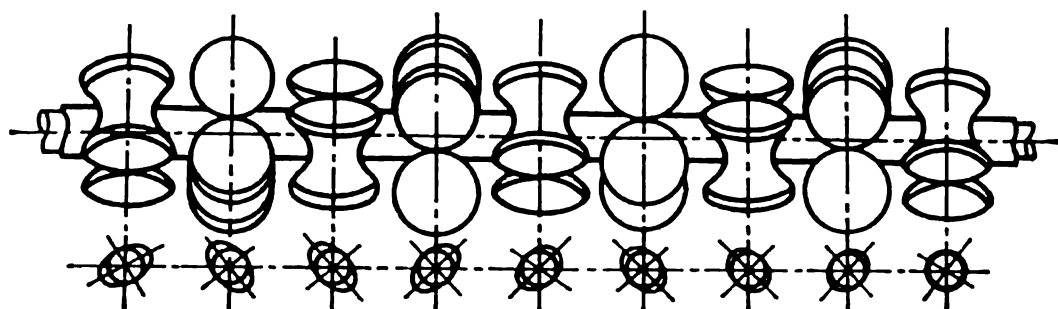


Fig. 114. Continuous seamless tube mill of recent design

*Furnace butt-welded pipe* is made by drawing skelp heated to 1300°-1350° C through a welding bell which bends the skelp along its longitudinal axis, forcing the edges into contact. At the point of contact the butted edges are welded together due to the pressure exerted by the bell. Tubes from 75 to 100 mm in diameter are produced by furnace butt welding.

*Electric resistance welding* enables tubing of higher quality to be obtained than by the furnace welding process. Tubing from 6 to 630 mm in diameter and with a wall thickness from 0.5 to 20 mm is produced by this method.

The initial material in the production of electric welded tubes is bright cold-rolled strip in coils, or sheet steel (for large-diameter tubing), which is preliminarily cleaned from scale and rust by pickling or shot blasting.

Electric resistance welding is the most widely used process in tube production but other electric welding methods are also applied. For example, submerged-arc welding is used for thick-walled tubes of medium size of carbon and alloy steels and large-size tubes of carbon steels; atomic-hydrogen welding for tubes up to 200 mm in diameter with walls from 2 to 12 mm thick of alloy steels; and argon-arc welding for thin-walled tubing of a diameter up to 450 mm with a wall thickness from 0.6 to 5 mm made of high-alloy austenitic steel, or of nonferrous metals and alloys.

A relatively new process of tube or pipe manufacture involves the forming of a strip into the tube by rolling it up along a helix

and then welding the helical seam by submerged-arc techniques. The product is known as submerged-arc welded helical-seam pipe.

Thin-walled tubes and tubes of small diameter are made by a cold-rolling process in special mills following hot rolling.

The metal consumption per ton of finished tubing or pipe varies from 1.05 to 1.18 tons depending upon the rolling process employed.

## Chapter 20

### METAL DRAWING AND EXTRUSION PROCESSES

#### 20-1. Metal Drawing Processes

*Metal drawing* is a process in which wire, bar stock or tubing is tension-drawn through the hole of a special tool, the drawing die, the hole being somewhat smaller in size than the initial material. This metal working process imparts accurate dimensions, specified cross section and a clean, smooth surface to the work. Finished products with very thin sections may be obtained by repeating the drawing operation several times.

Drawing, usually in the cold state, is applied to nonferrous metals and alloys; low-carbon, high-carbon and alloy steels; and alloys with special properties.

This operation is performed on powerful drawbenches with a high pulling capacity.

**Drawing dies and types of drawbenches.** The die, the main unit of a drawbench, consists of two component parts—the die proper *1* and the holder *2* (Fig. 115). The die hole is divided into four zones: deforming *I*, lubricating *II*, sizing *III* and outlet *IV*. The sizing zone is usually of a cylindrical form; the other zones are conical. The taper angle  $\beta$  of the deforming zone is selected from  $10^\circ$  to  $24^\circ$  depending upon the type of work and metal being drawn.

Dies are made of cemented carbides, black diamonds (for fine wire) or tool steel (for drawing rods and tubes of large cross section).

The end of the rod is pointed before drawing so that it freely enters the die hole and sticks out behind the die. This end is gripped by the jaws of the pincer, or carriage, which pulls the rod through

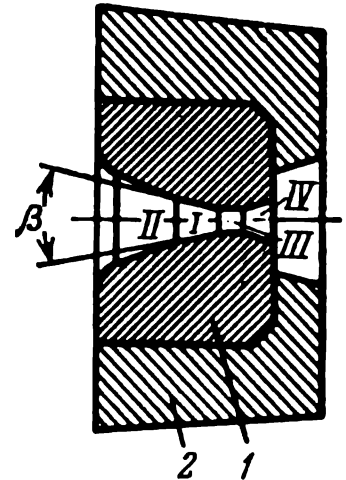


Fig. 115. Cross section of a rod or wire drawing die

all the zones of the die hole where it undergoes deformation—reduction and elongation.

According to the pulling method employed, drawbenches are classified as straight, or chain, type drawbenches and continuous bull-block machines.

In *chain drawbenches* the required sections are drawn in the form of long rods (3 to 5 m and longer). These drawbenches may have a pulling capacity from 1 to 150 tons; the drawing speed may reach 130 m per minute. The rod is drawn through the die with the aid of a motor whose rotary motion is converted by a system of gearing into straight-line motion of a chain and the pleyer (drawing carriage). The front end of the work, protruding from the die, is gripped by the jaws of the pleyer and the work is drawn through the die so that it acquires the specified cross section.

*Bull-block drawbenches* are employed in cases when the material being drawn and the product obtained can be wound on reels (wire, etc.). The process of drawing the work through the die in this case is also powered by an electric motor which drives the drawing block on which the wire or other similar product is wound up.

Bull-block machines may be either of the single- or multiple-die type; their application being determined by the required number of passes. The diameter of the drawing block in machines of various sizes may vary from 150 to 1000 mm; the available power ranges from 7 to 150 hp.

**Preparing the metal for drawing.** Before drawing the rod is cleaned of scale. Descaling is commonly done by pickling in acid solutions.

After pickling, the metal is washed to remove all remains of acid and sludge from its surface.

If the steel rod is to be subjected to several consecutive drawing passes, a sublayer for the lubricant is applied by an additional operation, such as yellowing, copper coating or phosphating followed by liming. Yellowing is accomplished by slight rusting of the bar in such a manner that a thin film of ferric hydroxide is formed on the surface. In copper coating the metal is dipped into a weakly acidified solution of copper sulphate. Liming is done by immersion for a short time in a boiling lime solution.

The final operation before drawing is drying at a temperature above 100° C. Drying drives off the moisture and a part of the hydrogen dissolved in the metal. This helps to avoid pickle brittleness of the metal.

A suitable lubricant is then applied to the dry surface of the rod. Lubrication reduces the required drawing force and energy consumption, enables a smooth surface to be obtained and increases die life.

Various kinds of mineral and vegetable oils, animal fats, graphite, soap and certain emulsions are applied as drawing lubricants.

**Metal drawing techniques.** The selection of the reduction value, drawing speed and kind of lubricant should be based upon the chemical composition of the metal, properties of the die material, and the shape and size of the cross section being drawn.

The coefficient of elongation  $\mu$  of the metal in drawing is equal to the ratio of the cross-sectional area  $F_1$  of the initial rod to the cross-sectional area  $F_2$  after drawing:

$$\mu = \frac{F_1}{F_2}. \quad (78)$$

The reduction  $\lambda$  in drawing is defined as the ratio of the difference in cross-sectional area before and after drawing to the initial cross-sectional area, expressed in per cent:

$$\lambda = \frac{F_1 - F_2}{F_1} \cdot 100. \quad (79)$$

As a rule, the coefficient of elongation in metal drawing is assigned within the limits 1.25-1.30 and the reduction may be up to 30-35 per cent.

Certain grades of cold-rolled stock, for example wire, bars, tubes, and strip must have a thin cross section and a fine finish. Such products undergo *sizing*, or *cold drawing*, i. e., they are drawn through a die of the proper size and shape without any appreciable reduction or elongation.

This cold-drawing procedure consists of the same operations, in the main, as those previously discussed. The reduction, however, is within 8 to 12 per cent since the chief aim of this operation is to maintain an accurate size and smooth surface, and not to increase the length of the work.

## 20-2. Extrusion of Metals

*Extrusion* is the process of forcing metal, enclosed in a container, through the opening of a die. In its cross section the extruded metal acquires the contour and dimensions of the die opening.

Extrusion is more widely used in fabricating nonferrous metals and their alloys, but steel and other ferrous alloys can also be successfully processed.

The initial material in extrusion is cast or rolled billets.

A great variety of shapes can be extruded (Fig. 116), including rods from 5 to 200 mm in diameter, tubing with an inside diameter up to 800 mm and a wall thickness from 1.5 to 8 mm, and many intricate shapes.

**Extrusion methods.** Two methods, direct and indirect extrusion, are employed.

In *direct extrusion* the billet 1, heated to a predetermined temperature, is placed in container 2 of the press (Fig. 117a). Die 4 with the required opening is secured on one end of the container by die holder 3. In the illustrated case the die has a circular opening. At the other end, ram 5 with dummy block 6 can enter the container. In operation, a plunger of the press transmits the required force to the ram which, through the dummy block, applies pressure on the billet. The metal is subjected to plastic deformation and is forced to flow through the die opening. At the end of the extruding operation a small piece of metal, called butt-end scrap, remains in the container and cannot be extruded.

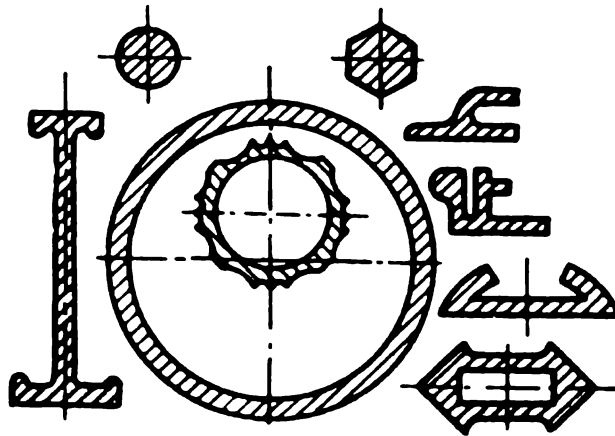


Fig. 116. Typical extruded shapes

In *indirect extrusion* (Fig. 117b) the extrusion die 4 is mounted on the end of a hollow ram 5 and enters container 2 instead of the dummy block. During the travel of the ram, the die applies pressure on the billet 1 and the deformed metal flows through the die opening in the direction opposite to ram motion. In indirect extrusion the amount of scrap is reduced to 5 or 6 per cent (it is about 18 to 20 per cent of the billet weight in direct extrusion) and the extruding force is less by 25 to 30 per cent.

Nevertheless, the indirect extrusion method finds only limited application due to the complex design of the press required.

In *tube extrusion* (Fig. 117c) the metal billet 1 in container 2 is first pierced by the mandrel 7. The front end of the mandrel passes through the whole billet and die opening, and projects a definite distance past die 4. This forms an annular clearance between the die opening and the mandrel. Upon travel of ram 5 together with dummy block 6 in the direction towards the die, the metal is forced through the annular clearance and forms tube 8. The outside diameter of the extruded tube equals the diameter of the die hole; the inside diameter equals that of the steel mandrel.

The yield in extruding usually varies from 70 to 80 per cent.

The metal is subjected to plastic deformation in the course of extrusion; it undergoes reduction and elongation.

The coefficient of elongation in extrusion equals the ratio of the cross-sectional area  $F_c$  of the container bore to the cross-sectional area  $F_d$  of the die opening:

$$\mu = \frac{F_c}{F_d} . \quad (80)$$

As a rule, the coefficient of elongation ranges from 8 to 50, but larger values are sometimes used.

The reduction of the metal in extrusion is defined as the ratio of the difference between the cross-sectional areas of the container and die opening to the cross-sectional area of the container. It is expressed by the equation

$$\lambda = \frac{F_c - F_d}{F_c} \cdot 100 \quad (81)$$

and may reach 90 per cent.

Friction between the container walls and the metal being extruded is reduced and a more uniform flow of material from the die is achieved by the application of a lubricant. Lubricants employed for this purpose include a mixture of machine oil and graphite, molten glass and other lubricating materials.

Metal extrusion operations are generally performed in hydraulic presses with either vertical or horizontal plungers. Mechanical presses are used to a lesser extent.

The power-transmitting fluid in a hydraulic extruding press is water. Water is delivered at a pressure of 200 to 400 kg per sq cm by a high-pressure pump to the press cylinder where it actuates the plunger. The plunger transmits the pressure to the ram and dummy block which force the metal through the die.

In a mechanical press the ram may be powered by a motor.

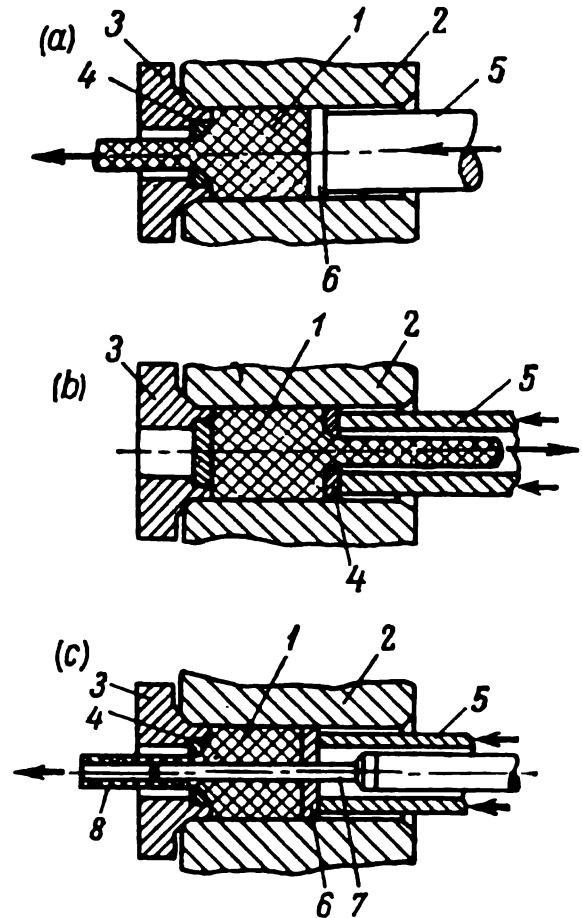


Fig. 117. Extrusion methods:  
(a) direct extrusion; (b) indirect extrusion; (c) tube extrusion



## Chapter 21

## THE FORGING AND PRESSWORKING OF METALS

The processes of reducing a metal billet between flat dies or in a closed impression die to obtain a part of predetermined size and shape are called *smith forging* or *closed-die forging*, respectively.

Various working and auxiliary tools are employed in forging processes.

By proper selection of the billet and the types of tools, and by suitably positioning the billet in respect to the tools, many kinds of parts (called forgings) may be produced.

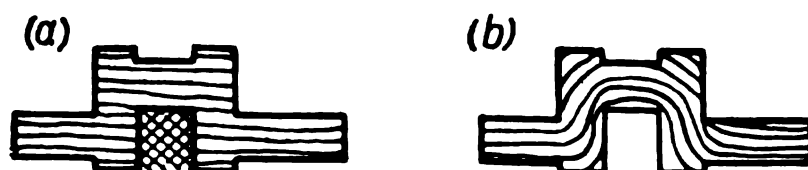


Fig. 118. Grain flow lines shown schematically in a crankshaft:

(a) machined from the solid; (b) drop forged

Forging, as well as other mechanical working processes, is associated with a change in the macrostructure of the metal that leads to a rearrangement of the fibres, and alters the grain size. Fig. 118 illustrates the fibre grain flow in metal as revealed by etching a macrosection of a crankshaft produced by two different manufacturing processes. The crankshaft shown in Fig. 118a was turned in a lathe from a solid blank; the one shown in Fig. 118b was made by closed-die forging. Fibre grain flow is much more favourable in the second case since it promotes a higher strength in the crankshaft.

Directional working of the metal characteristic of most forging processes enables parts to be produced with accurate dimensions, a smooth surface and sufficiently high mechanical properties.

### 21-1. Smith Forging

Smith forging, also called flat-die and open-die forging, usually requires preheating of the metal.

Smith forging performed in a press or hammer is called *press* or *hammer forging*.

Smith forging, done by hand on an anvil, is employed only to shape a small number of light forgings, chiefly in repair shops. Heavy

forgings weighing up to 350 tons, as well as medium forgings in small batches, are produced exclusively in hammers and presses.

The initial material in smith forging may be ingots, blooms or rolled billets of various cross sections and lengths.

The billet weight is determined from the following formula on the basis of the weight of the forging:

$$Q_b = Q_f + Q_c + Q_s + Q_t \text{ kg,} \quad (82)$$

where  $Q_f$  = weight of the forging,  
 $Q_c$  = weight of the cropped ends,  
 $Q_s$  = weight of the scale loss,  
 $Q_t$  = weight of trimming scrap.

In forging ingots the weight of the cropped ends is from 25 to 30 per cent of the ingot weight. Scale loss is taken as 2 or 3 per cent of the ingot or billet weight for each heating and from 1.5 to 2 per cent for each subsequent reheating. The weight of the trimming scrap depends upon the complexity of the forging and the processing method employed; for forgings of simple form it ranges from 5 to 8 per cent, for certain intricate forgings it may reach 30 per cent of the billet weight.

The next step is to establish the shape and dimensions of the billet, proceeding from the drawing of the forging which makes provision for processing allowances (machining, descaling, etc.) and extra stock to simplify the shape of the forging. Then a suitable hammer or press is selected, one capable of performing the required forging operations. These operations must provide for good working of the metal throughout its cross section.

The degree of working is commonly evaluated by the coefficient of forging reduction which is the ratio of the cross-sectional area of the billet to that of the finished forging. This coefficient should be at least 3 to 5 for steel ingots and from 1.1 to 1.5 for rolled billets.

**Smith forging operations.** The principal operations in smith forging are: upsetting, drawing down, fulling, cutting off, punching or piercing, expanding and offsetting.

In *upsetting* (Fig. 119a) the cross-sectional area of the billet is increased due to a reduction in its height. By repeated upsetting from different sides the billet can be returned to its initial shape but the metal will be of higher quality and its properties will be uniform in all directions.

The coefficient of forging reduction in upsetting in a single direction is calculated as the ratio of the initial to the final height or as the ratio of the final to the initial cross-sectional area.

Upsetting, involving only a part of the billet, is called *heading* (Fig. 119b). Heading may be done by heating only a part of the billet (the end or the middle) or by restricting deformation of a section of the billet by means of a ring-shaped tool  $K$ .

In *drawing down* (Fig. 119c) the length of the billet is increased by reducing its cross-sectional area. The billet may be drawn down beginning either at the end or from the middle.

If, in drawing down, the billet is turned  $90^\circ$  so that the blows of the tool are applied to both sides of the billet, the operation is called *drawing down with turning*.

The billet can be more intensively drawn down if the spread is decreased. This can be done by reducing the feed  $a$ . In practice the rate of feed  $a$  is from 0.4 to 0.75 of the width  $B$  of the flat die.

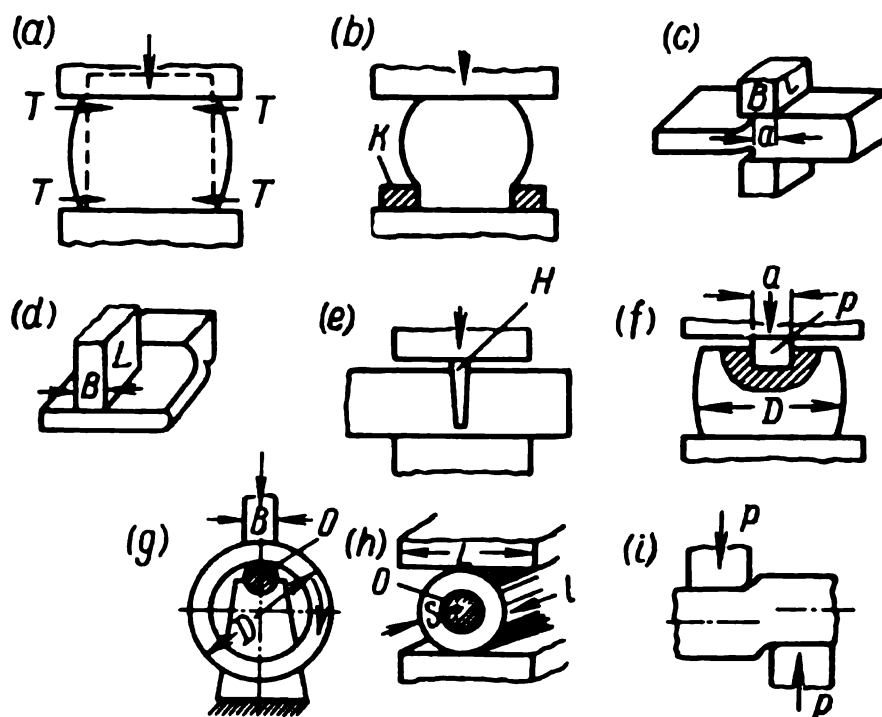


Fig. 119. Smith forging operations

*Fulling*, or spreading, the metal along the length of the billet (Fig. 119d) is done by working separate sections; in this case, the axis of the billet is positioned perpendicular to the width of the flat die.

The billet can be cut up into several pieces by *cutting off* (Fig. 119e), an operation done with a suitable tool  $H$  (chisel or hardy).

*Piercing* (Fig. 119f) with a punch  $P$  is employed to obtain blind or through holes in the metal. Pierced billets may be further

(1) *expanded* on their diameter  $D$  (Fig. 119g) by using a mandrel  $O$ , thereby reducing the wall thickness of the ring and increasing its diameter (the longer axis of the flat die is parallel to the mandrel axis), or they may be

(2) *drawn down* along the hole axis (Fig. 119h), in which case the length of the ring is increased by reducing the wall thickness (the longer axis of the flat die is perpendicular to the ring axis).

Among the more complicated forging operations is *offsetting* the metal of the billet (Fig. 119i).

**Power hammers and presses for smith forging.** Small parts are usually smith forged in pneumatic power forging hammers, large parts in air-and-steam hammers and very heavy parts in hydraulic presses.

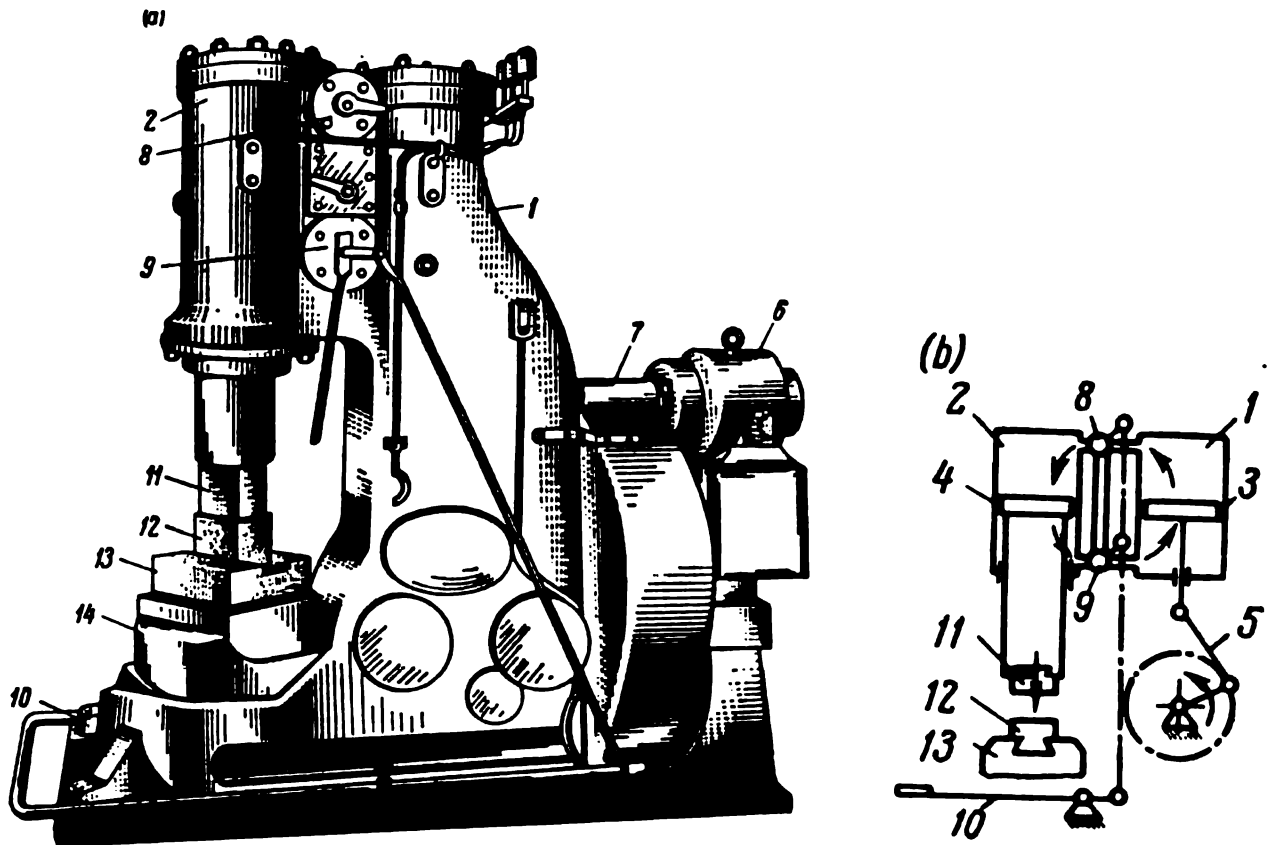


Fig. 120. Pneumatic forging hammer:  
(a) external view; (b) gearing diagram

A *pneumatic forging hammer* (Fig. 120) has two cylinders: compressor cylinder 1 and ram cylinder 2. Piston 3 of the compressor cylinder compresses air and delivers it to the ram cylinder where it actuates the piston which is in fact integral with ram 4 delivering the forging blows to the work. The reciprocation of the compressor piston is obtained from a crank drive 5 which is powered from motor 6 through reducing gear 7.

The air distribution device between the ram and compressor cylinders consists of rotary valves 8 and 9 with ports through which air passes into the ram cylinder, below and above the piston, alternately. This drives the ram up and down, respectively. The valves are switched over by depressing foot treadle 10 or operating a hand lever.

By controlling air distribution the required ram movement can be attained: either continuous blows are delivered to the forging.

or the forging is held down, or squeezed, on the anvil, or the ram is held in its upper position.

The ram die 11 is secured to the ram and the anvil die 12 to cap 13 by means of taper keys in a dovetail joint.

The steel cap 13 is mounted on the heavy anvil 14 which is separate from the hammer frame.

The size of a pneumatic hammer, characterised by the falling weight (weight of the ram piston, ram and ram die) may vary in a range from 50 kg to 1 ton. The weight of the anvil is commonly from 15 to 20 times the falling weight. Hammers operate at 70 to 190 blows per minute.

*Air-or-steam hammers* can be operated by either steam or compressed air. They do not have a built-in compressor and, therefore, require additional facilities for supplying high-pressure steam or compressed air.

As to their principle of operation, these hammers are classified as (1) single-action and (2) double-action types.

In the first type the air or steam is required to raise the ram, rod, die and piston which then fall under their own weight to deliver a blow. Such hammers are almost obsolete at present.

In double-action hammers the steam or air serves both to raise the ram and to increase the energy of the blow on the down stroke.

An arch-type air-or-steam forging hammer is illustrated in Fig. 121.

Steam or air is admitted to the cylinder through slide valve 1 which is controlled by the operator with lever 2. The ram die 3 is secured to the dropping ram 4 in the ordinary manner. The ram is fastened to the lower end of rod 5 whose upper end is secured in piston 6 in the cylinder.

Anvil die 7 is secured in the steel cap 8 and the latter in the heavy anvil 9.

The air-or-steam hammer can operate with either single strokes or continuous blows of the ram die on the forging. If required, the forging can be held down (squeezed) to the anvil or the ram can be held up.

Such hammers are available with falling weights from 0.5 to 5 tons. The required air or steam pressure is from 6 to 8 kg per sq cm.

*A hydraulic forging press* is shown schematically in Fig. 122.

The cylinder 1 of the hydraulic press is mounted in the upper platen 2 which is tied by columns 3 to the lower platen, or bed, 4 erected on the foundation. The ram 5 travels in the cylinder and is firmly attached to the moving slide, or presshead, 6. The latter carries the upper die 7 while the lower die 8 is secured to the bed.

Downward travel of the slide with the die, i. e., the working stroke of the hydraulic press, is due to the pressure of ram 5 which, in turn, is actuated by the working fluid. The slide is returned to its top position by the action of the working fluid on rams 9 in the pull-back

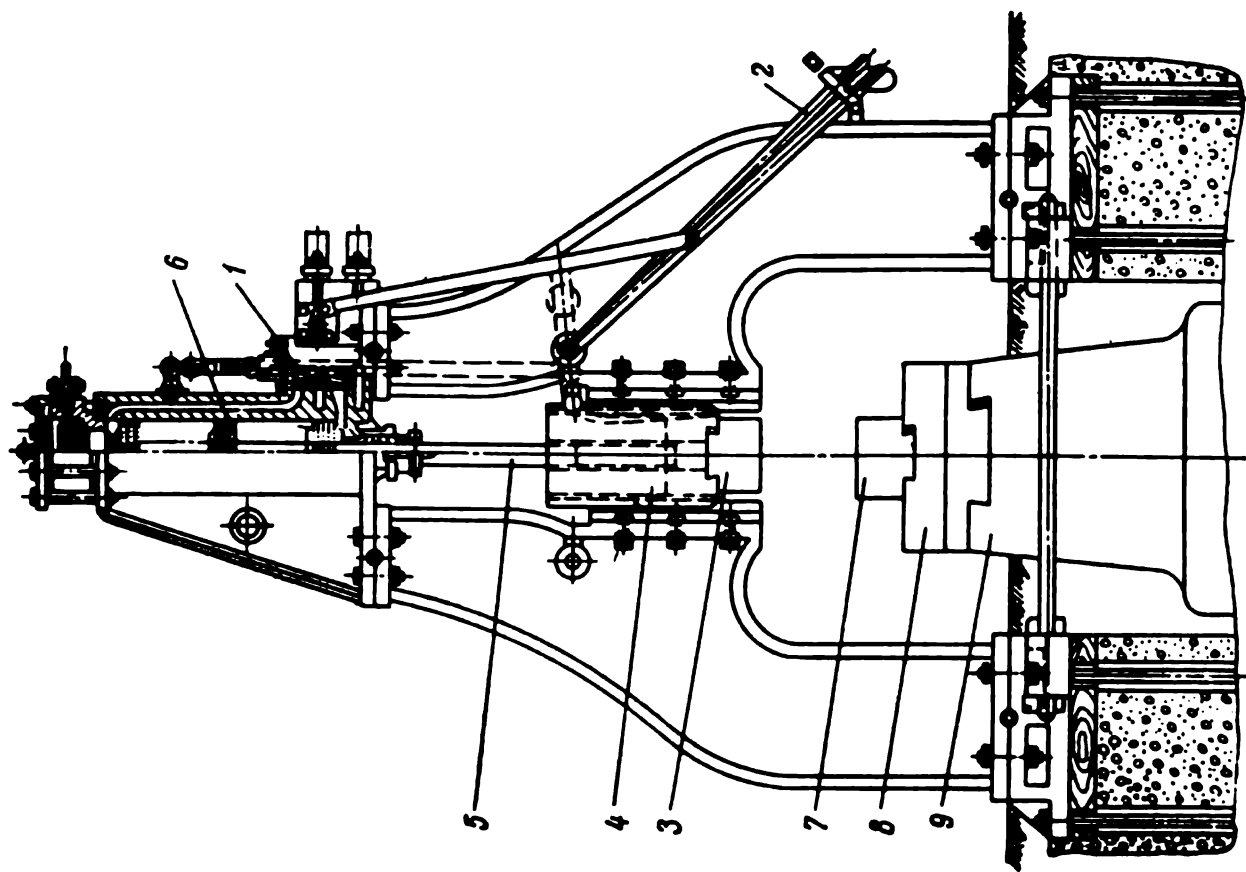


Fig. 121. Arch-type air-or-steam forging hammer

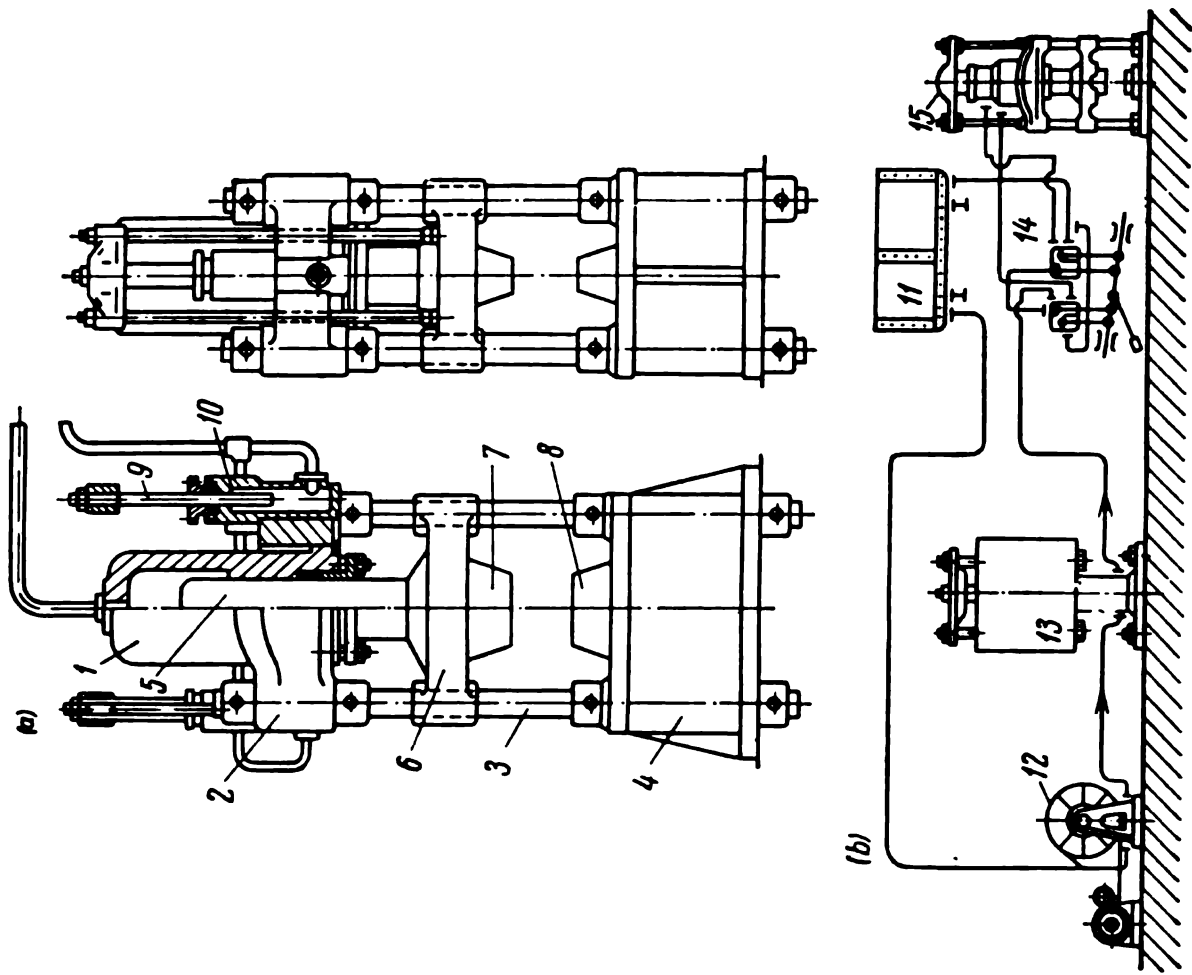


Fig. 122. Hydraulic forging press:  
(a) press proper; (b) hydraulic system

cylinders 10. The press controls are arranged on the distributor which has a control lever and by-pass valves.

Water, certain types of emulsion, or mineral oil at a pressure up to 300 kg per sq cm is used as the working fluid. The fluid passes first from the large-capacity tank 11 to pump 12 and then is delivered to the press 15 with the aid of accumulator 13 and distributor 14. The fluid accumulator enables a pump with a lower delivery to be used or to supply the working fluid simultaneously to several presses in amounts exceeding the capacity of the pumps.

Hydraulic forging presses can develop a force between the dies ranging from 300 to 10,000 or 15,000 tons.

## 21-2. Hot Closed-die Forging Processes

Forging hot metal in impression dies includes such processes as closed-die or drop forging, press forging, and machine, or upset, forging. It must be distinguished from the pressworking of sheet metals which is usually done in the cold state.

In *closed-die forging* the flow of the metal being worked is restricted by the shape of impressions in a die manufactured beforehand for the given job. More often than not the metal is suitably heated before forging in which case it is called a *hot forging* process.

The dies used in drop forging consist of two mating halves which, when assembled, form one or several internal cavities, called *die impressions*.

So-called drop forgings, obtained by this process, can be produced in a wide range of sizes and shapes. Drop forging is most commonly employed for the mass or large-lot production of parts of steel or non-ferrous metals of a weight up to 350 kg.

**Closed-die forging procedures.** The initial materials for hot closed-die forging are rolled bar stock of round, square, rectangular and other cross sections, certain die-rolled shapes and other blanks and slugs.

Two general drop forging procedures are ordinary forging and flashless forging. In the first case the forging obtained has a flash, or fin, around the parting line and the cavity in the die is properly filled with metal since the volume of the slug or blank is somewhat larger than that of the die cavity. No flash is provided for in flashless forging as there is no excess metal since the volumes of the slug and cavity are equal. At the present time, the ordinary method of forging with a flash is more extensively employed in spite of the metal loss in the trimmed off flash.

Drop (closed-die) forging may be performed in either single- or multiple-impression dies. The single-impression type is suitable for forgings of simple shape, and the size and shape of the impression must

correspond to those of the finished hot forging. Forgings of irregular shape are made in multiple-impression dies which contain preliminary impressions in addition to the finish impression. In some cases the preliminary and finish impressions are arranged in two or more separate dies (combination drop forging process).

Forging blanks heated to the forging temperature can be drop forged in power hammers and crank friction presses.

**Closed-die forging in drop hammers.** The most up-to-date procedure in this line is considered to be single- or two-impression closed-die forging in hammers with preliminary preparation of shaped for-

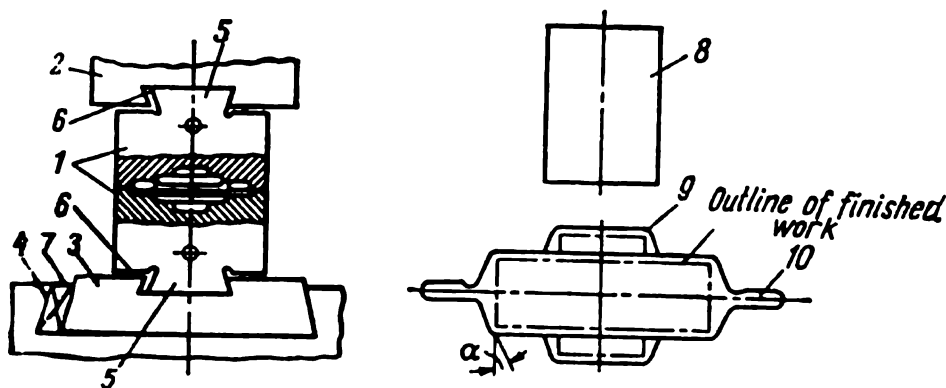


Fig. 123. Production of a simple forging with flash

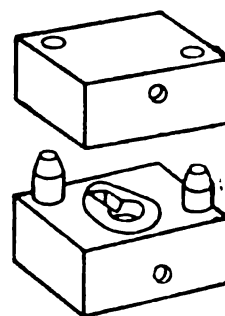


Fig. 124. An unattached die

ging stock by ordinary or die (periodic) rolling. In regular practice, however, multiple-impression closed-die forging still finds widespread application.

Fig. 123 illustrates schematically the production of a simple forging with flash from a cylindrical slug in a single-impression die. The die 1 consists of two halves, the top half being fastened securely in the ram 2 and the bottom half in die-holder 3 fastened, in turn, in the anvil. Fastening facilities include dovetail 5 and keys 6 and 7. The impression is in both halves of the die.

Draught must be provided on all the side surfaces of the forging ( $\alpha=3^\circ$  to  $15^\circ$ ) to facilitate its removal from the die.

The preheated slug 8 is placed on end into the bottom half of the die. Upon the blows of the top die half on the slug, the metal fills the impression, forming the forging 9.

The most difficult parts of the impression to fill are the corners. When the distance between the halves of the die is sufficiently small, the flash 10 that is formed prevents further flow of metal from the impression and the metal is forced to fill all the corners in the impression. The flash, or fin, is subsequently cut away in a trimming die.

Unattached dies (Fig. 124) are sometimes employed to forge simple work. These dies, as their name implies, are not attached to the hammer ram and anvil. Forging is done by the blows of the flat ram die



on the top half of the unattached die. Guide pins are provided in the die to properly align the top and bottom halves.

A multiple-impession die and the forging sequence for a connecting rod are shown in Fig. 125.

The heated forging stock is first placed into the fullering impression 1 where it is reduced by the blows of the hammer, i. e., elongated by decreasing the cross section. The next operation, in the edging impression 2, redistributes the metal; the cross section is increased at

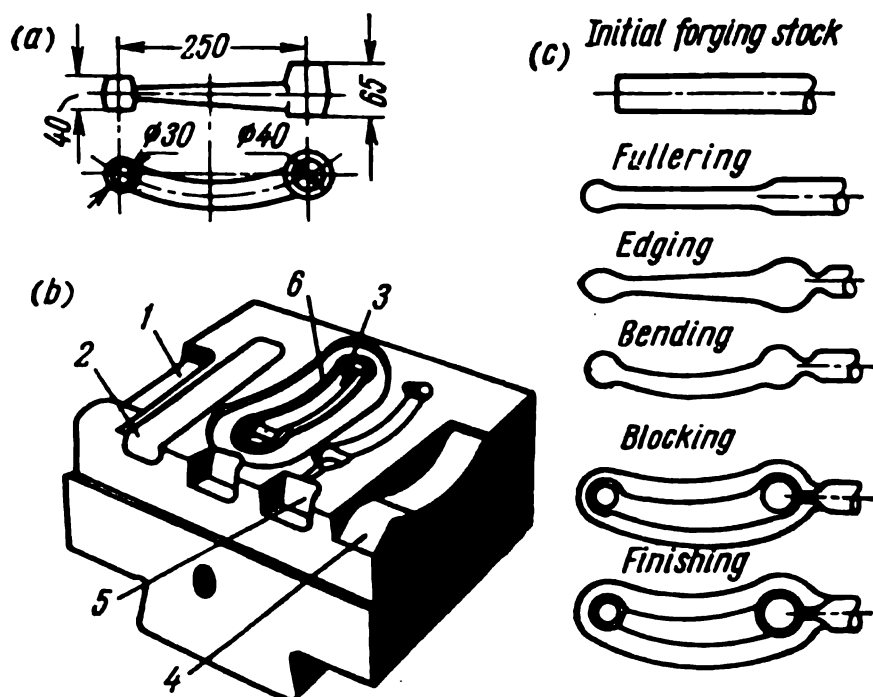


Fig. 125. Multiple-impession forging die:  
(a) part drawing; (b) bottom half of the die; (c) forging sequence

certain places and reduced at others as required to fill the cavities of the die. Before being forged in the finish impression 3, the stock is processed in the bender 4 and the semifinish (blocking) impression 5. By giving the forging its general shape, the blocking operation reduces the wear of the finish impression. The finish impression imparts the final shape to the forging. The flash gutter 6 is provided only around the finish impression.

Forging in the blocking and finish impressions is done in one (less frequently) or several blows of the hammer ram.

Air-or-steam drop hammers with a falling weight from 0.5 to 30 tons are the equipment used, in the main, for closed-die forging. Board drop hammers with a falling weight from 0.5 to 2 tons are employed to a lesser degree.

*Air-or-steam drop hammers* have much longer ram guides than those of forging hammers. Moreover, they have heavier anvils and more rigid frames. These features ensure higher forging accuracy.

**Press forging.** The highly efficient production of die forgings in *mechanical forging presses* of the crank type has found wide application in forging practice of the last 25 years.

The character of metal deformation in press forging substantially differs from that of hammer forging. One blow of the hammer works the metal only in the surface layers of the forging so that deformation does not effectively penetrate into the volume of the metal. The push or squeeze pressure of a press applied to the stock gradually increases and penetrates deep into the metal, involving its whole volume. This feature of press forging is taken into consideration in designing die equipment.

Forging dies for crank presses are designed so that the forging is formed in several operations in preliminary impressions. At the same time, the number of impressions is held to a minimum. It is good practice to employ shaped forging blanks (for example, die-rolled stock).

It is necessary to remove the scale from the heated blank before inserting it into the die since otherwise the scale may become embedded in the surface of the forging. Hydraulic descaling is practised in many plants for this purpose. It consists in processing the heated blank with a stream of water at high pressure (up to 100 kg per sq cm). The water cools the scale, breaks it up into pieces and blows it off.

Both ordinary and flashless forging may be done in presses. Forgings may also be produced by squirting techniques.

As an example, Fig. 126 illustrates the die forging of a gear blank with flash in a crank press. The die set (Fig. 126a) used for this purpose has two interchangeable impression-die inserts 1 and 2. These inserts are rigidly attached to the upper and lower shoes 3 and 4 by means of strap 5 and screws 6. Alignment of the die inserts is maintained by one to four guideposts 7. Knockouts 8 and 9 push the finished forging out of the die.

The gear blank is forged in the following manner. After upsetting the stock in the first impression (Fig. 126b) it is blocked in the second impression (Fig. 126c). The forging acquires its final form in the finish impression. The forging has a flash that is subsequently trimmed in another die.

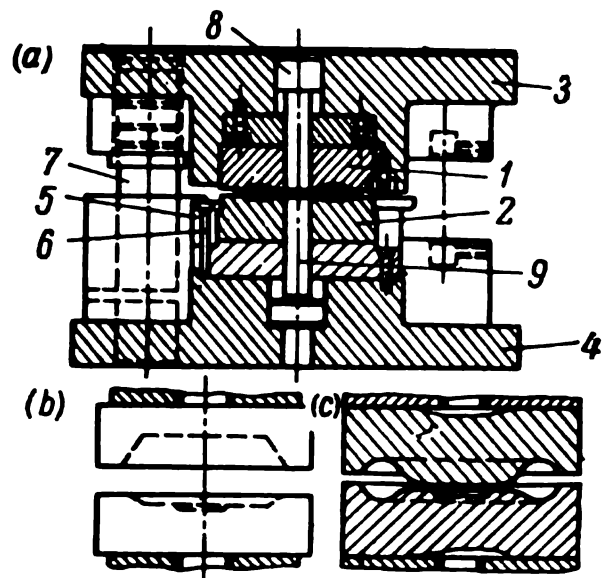


Fig. 126. Die set for forging a gear blank

Flashless forging in a die set is shown in Fig. 127. The heated slug is upset in the first impression (Fig. 127a), the recess is formed in the second impression (Fig. 127b), while the forging obtains its final shape in the third impression which is of the enclosed, flashless type (Fig. 127c).

A schematic view of a hot forging crank press and its gearing diagram are shown in Fig. 128. The operative units of the press are powered from motor 1 mounted on the press frame. By means of the V-belt

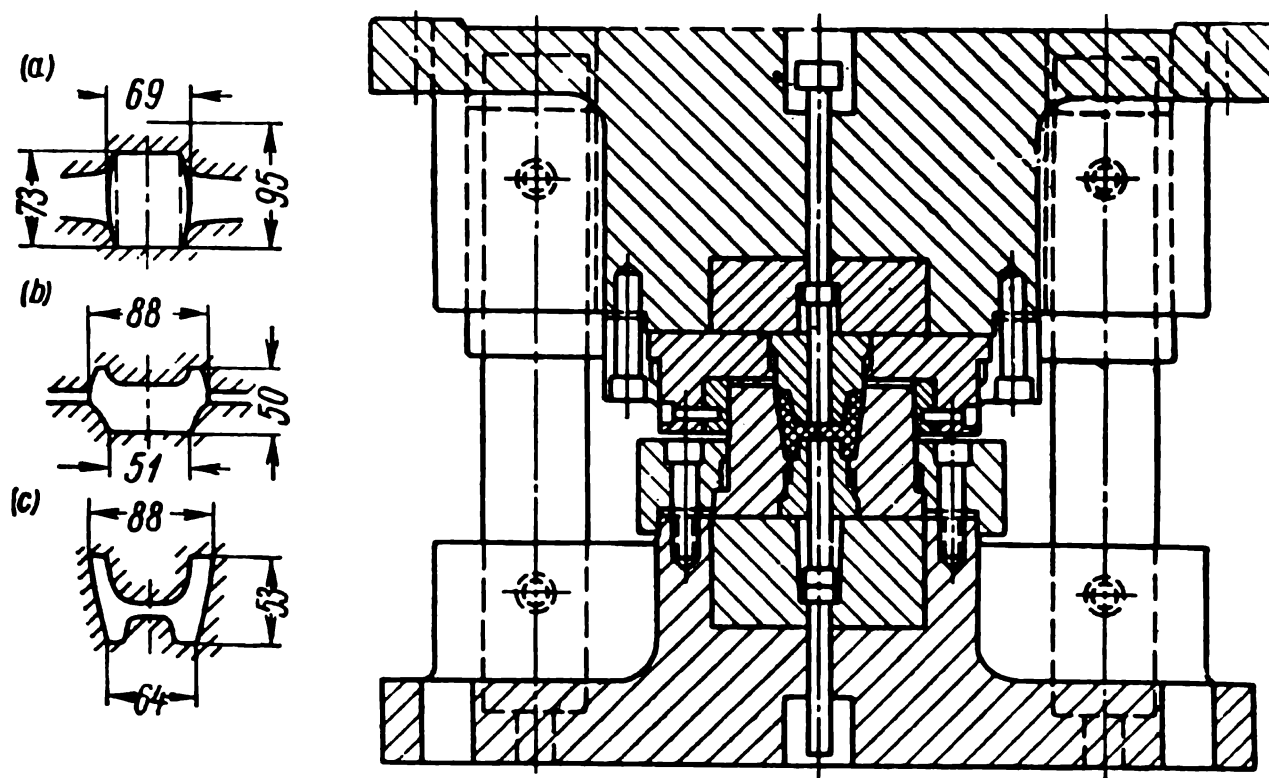


Fig. 127. Die set for flashless forging

drive, power is transmitted from pulley 2 to flywheel 3 mounted on auxiliary shaft 4. Flywheel 3 is equipped with a friction safety device to prevent the overloading of shaft 4. The flywheel is stopped by the auxiliary brake 5 which is automatically engaged when motor 1 is switched off. On the other end of shaft 4 gear 6 is mounted. It meshes with gear 7 which drives crankshaft 8. The latter, through pitman 9, reciprocates slide 10. The crank mechanism is engaged by pneumatic clutch 11 which is controlled by a pedal. Band brake 12 stops the crankshaft and slide when the pneumatic clutch is disengaged. The top half of the die is secured to the slide and the bottom half to the bed which is provided with a double-wedge device 13 for adjusting the die space.

Crank-type hot forging presses are distinguished for the high strength and rigidity of their frames. They are available with rated capacities from 500 to 8000 tons and speeds from 35 to 90 strokes per minute.

The size (capacity) of press required for producing a certain forging can be determined approximately by multiplying by 1000 the falling weight of the hammer needed to forge the same work.

Hot closed-die forging, or its separate component operations (bending, fullering, piercing, etc.), can also be carried out in hydraulic or friction presses.

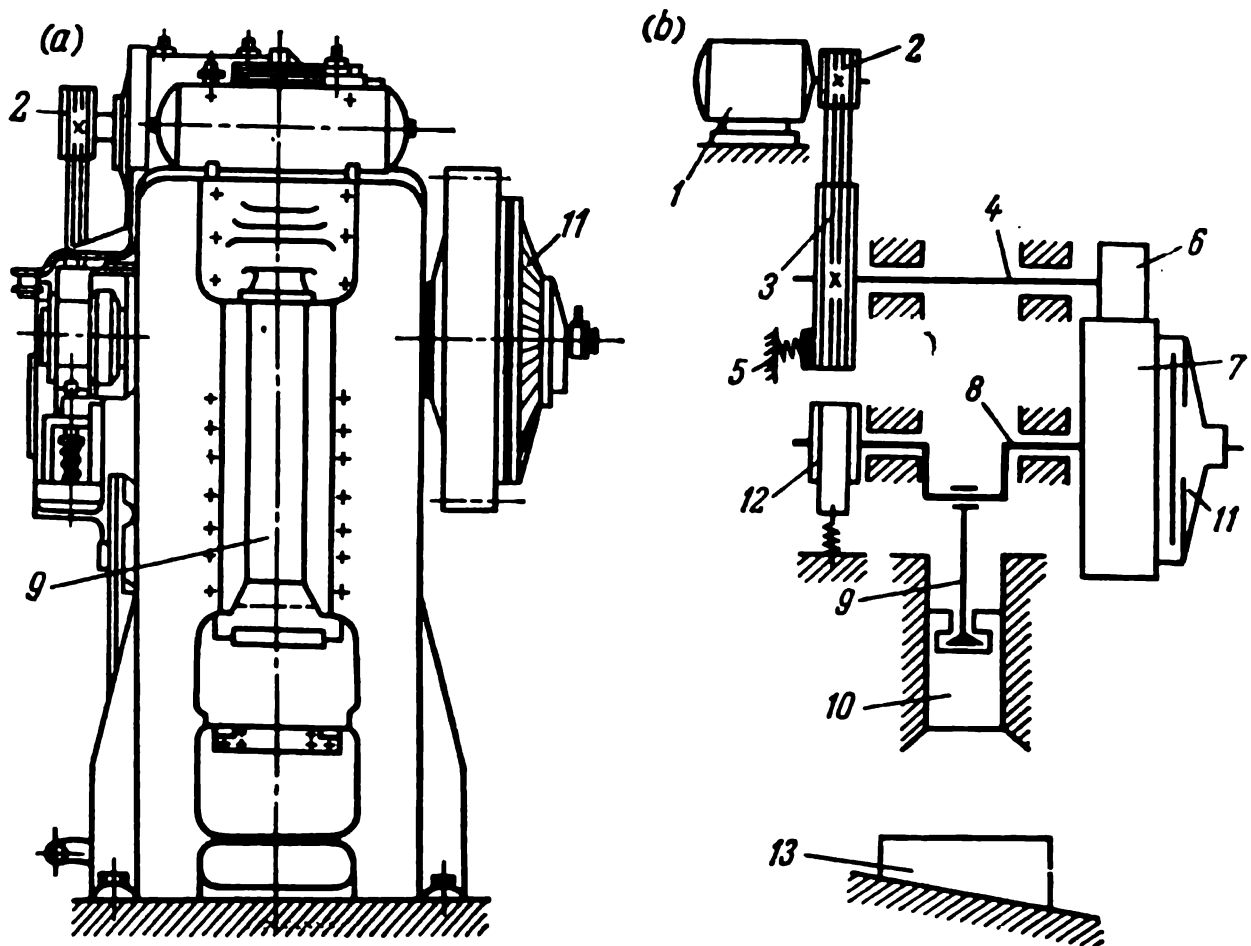


Fig. 128. Hot forging crank press:  
(a) general view; (b) gearing diagram

Hydraulic presses are employed in the production of heavy forgings weighing from 100 to 350 kg and more. A new technique recently applied for hydraulic presses is forging the blank in parts (sectional forging).

Friction screw presses are used for the most part to produce small forgings.

**Machine, or upset, forging.** Forgings of the ring and rod types with all kinds of heads and shoulders, such as bolts, nuts, washers and collars, can be conveniently produced in horizontal forging machines.

Typical of machine forging is the use of multiple-pass die sets consisting of punches and split dies, and the fact that the forging is made of bar stock.

A forging sequence for a horizontal machine is illustrated in Fig. 129. The heated end of the bar stock is inserted into the stationary die 1 up to stop 4. At this time punch 3 is in its left-hand position (Fig. 129a). Next, the movable die 2 grips the bar stock and, at the same time, a recess (impression) is formed in the closed dies for shaping the projecting stock; stop 4 is automatically retracted to its idle position (Fig. 129b). Then the punch advances to upset the bar end and forms the finished forging (Fig. 129c) and finally, the movable die and the punch are withdrawn to their initial positions (Fig. 129d). The rod and forging are shifted to the next pass where the forging is cut off.

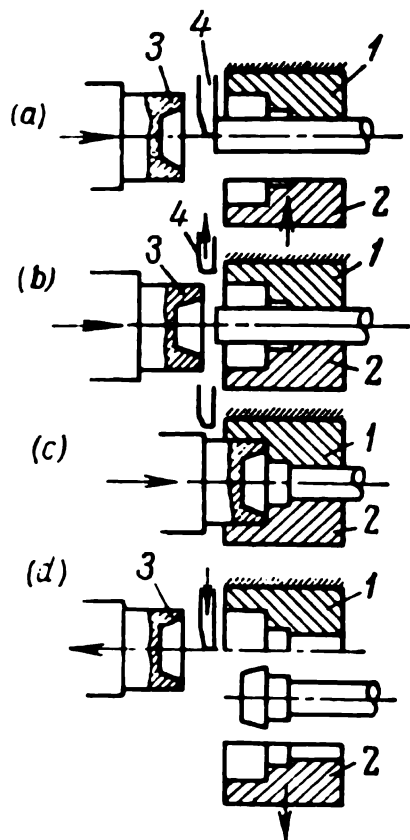


Fig. 129. Sequence of operations in a horizontal forging machine

Die sets for horizontal forging machines are designed with the required number of passes and punches. It is necessary to see that the bar stock is not bent axially in the forging operation since this will lead to the formation of a one-sided flash and seams in the metal. A set of split dies and punches for a machine forging operation are shown in Fig. 130. Several passes of consecutively larger cross section may be provided in the die to make a forging with an upset end.

The external view and gearing diagram of a horizontal forging machine are illustrated in Fig. 131. Through V-belts 2 motor 1 drives flywheel 3 which is engaged through clutch 4 to auxiliary shaft 5. Power is transmitted from the latter through gears 6 to main crankshaft 7 whose crank is linked by pitman 9 to the main, or heading, slide 8. The main crankshaft is also linked through two eccentrics 10 to the side, or gripping, slide 11 by means of a pitman and toggle lever system 12.

The die set consists of the stationary multiple-pass die 13 secured in the frame of the machine, movable multiple-pass die 14 attached to the gripping slide 11 and the upsetting punches 15 which are attached to the heading slide 8 and located in line with the impressions in the dies.

In operation, the heated bar stock is inserted into stationary die 13 and advanced to stop 16 which is linked to roller 17. The latter lies on a guide attached to heading slide 8. In the forward stroke of heading slide 8, a projection on the guide raises roller 17, thereby turning the stop shaft and withdrawing the stop from the line of travel of heading punch 15. At the same time movable die 14 is ad-

vanced by the gripping slide to match the stationary die 13 and to form the die impression. Punch 15 upsets the metal to fill the impression, thus producing the required forging.

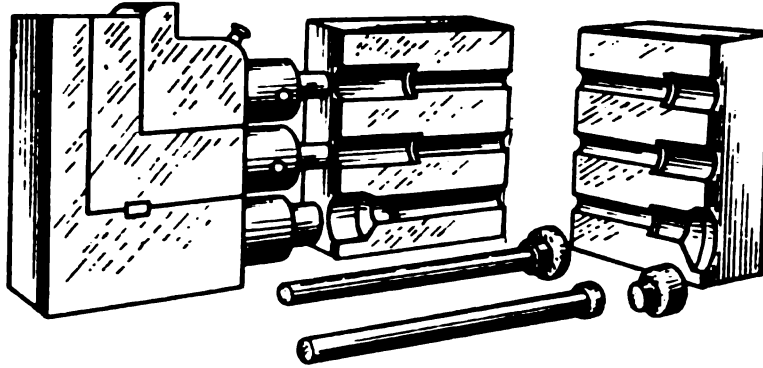


Fig. 130. Set of split dies and punches for a machine forging operation

Horizontal forging machines operate at a high rate; they have an output of from 400 to 600 forgings per hour and develop a forging force of from 100 to 3000 tons.

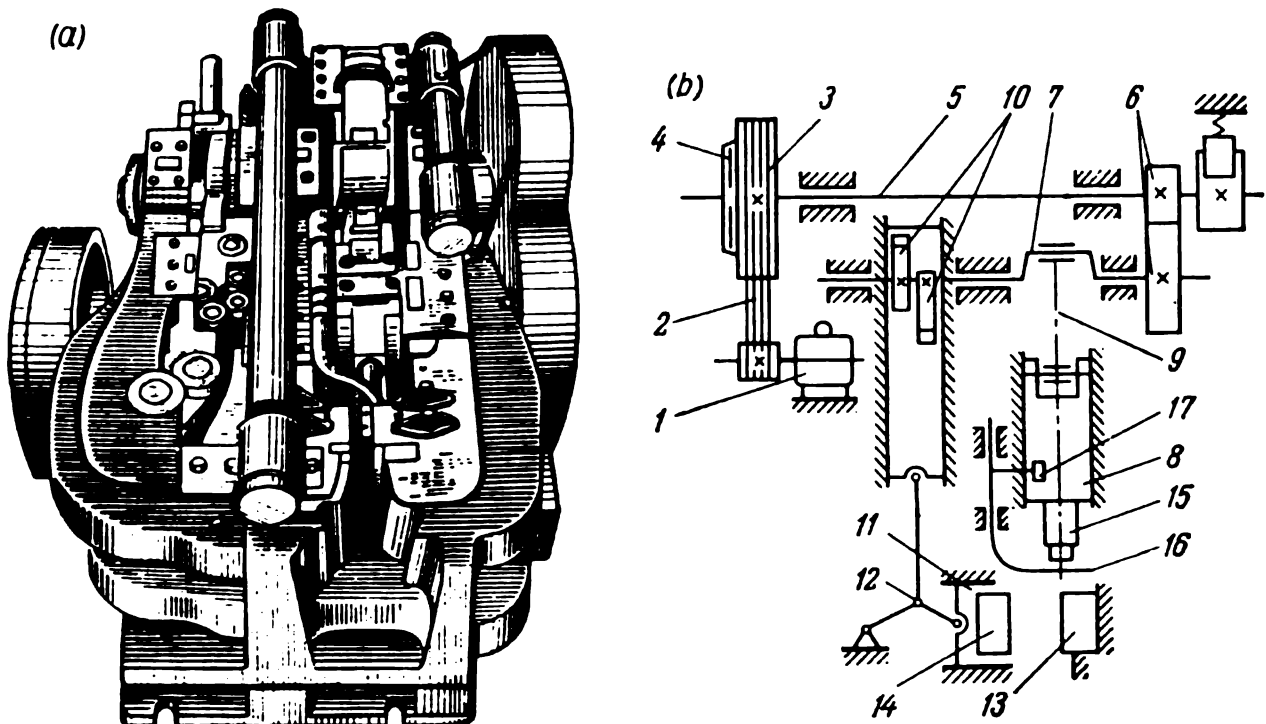


Fig. 131. Horizontal forging machine (model ГKM):

(a) external view; (b) gearing diagram

The capacity of the forging machine required for a given job can be approximately determined from the formula:

$$P = K\sigma F \text{ kg}, \quad (83)$$

where  $K$  = correction factor, equal to 4 on an average,  
 $\sigma$  = tensile strength of the metal at the final temperature in the forging operation, in kg per sq mm,

$F$  = projected area of the forging on a plane square to the direction of punch motion, in sq mm.

Horizontal forging machines accommodate blanks or stock from 25 to 300 mm in diameter.

*Forging rolls* can be efficiently used for the preliminary reduction of forging stock as a preparation for subsequent die forging, as well as for the production of certain straight and tapered forgings (of the arrow or spear type).

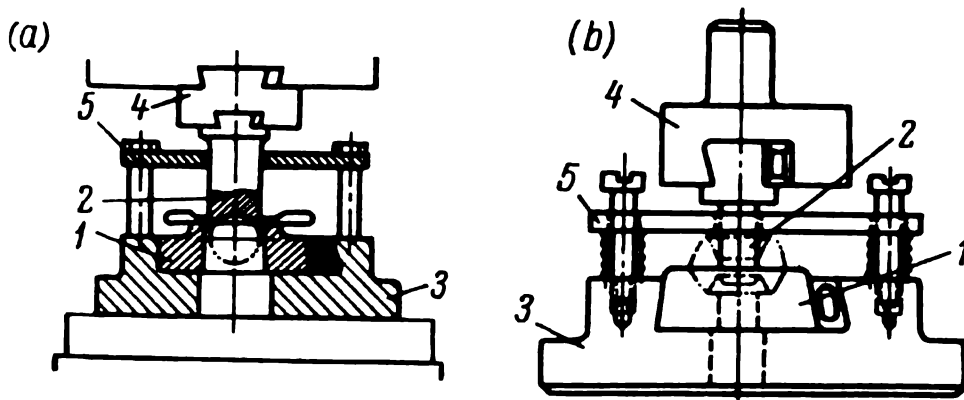


Fig. 132. Types of trimming dies:

(a) for cutting away the flash; (b) for punching out slugs

**Finishing operations on hot closed-die forgings.** The flash, or fin, formed in ordinary die forging in hammers and presses, is cut away by a trimming die in a crank press.

If forgings are made for parts with holes the slugs of metal in the holes are punched out in the trimming operation.

Dies for cutting away the flash and for punching out slugs are shown in Fig. 132. The die, in both cases, consists of die block 1 and punch 2. In the flash trimming die the cutting edge is on the die block; in the punching-out die it is on the punch. The other parts of the dies are designed either for pushing or supporting the forging and are shaped to fit the forging at the place of contact.

Die block 1 is secured by a tapered key to shoe 3 which is mounted on the bed of the press. Punch 2 is fastened in upper shoe 4 which is clamped by its shank in the slide of the press. Stripper 5 serves to push the cut-off flash or the punched-out forging off the punch.

Trimming may be done either hot or cold. Hot trimming is performed directly after the forging operation on a press included in the forging setup. Burrs formed when the flash is trimmed are removed by snagging on a grinder.

After cleaning up the forging in accordance with the established operation sequence it undergoes heat treatment, if necessary, straight-

ening in special dies in a hammer or press and a cleaning operation to remove forging scale. The latter is done either by barrel tumbling, sand or shot blasting in suitable equipment or pickling in an acid solution.

Sizing and coining are employed to impart a more accurate shape and size to the whole forging or to certain of its parts, and also to improve the surface finish.

### 21-3. Cold Die Forging Processes

Various cold forging techniques are employed in forging practice. Compression, or squeezing, operations usually include plane sizing, three-dimensional sizing (swaging), forging by the coining process and certain other specific operations.

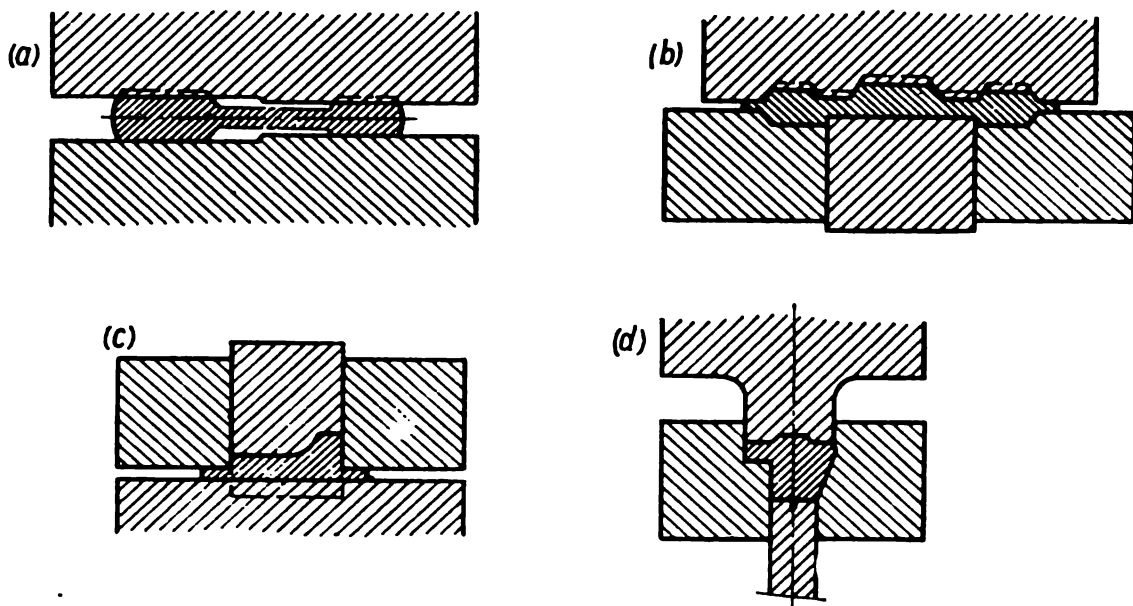


Fig. 133. Compression or squeezing operations

*Plane sizing* (Fig. 133a) enables accurate dimensions to be obtained; it straightens the forging and improves the surface finish. *Three-dimensional sizing*, or swaging (Fig. 133b) imparts the final size to the whole forging or to any of its parts. As a rule, this operation is done in a die where a flash is formed that is subsequently trimmed. *Forging by the coining process* is done either with or without the formation of a flash. Fig. 133c shows such an operation in a die in which a part of the metal forms a flash. This is often referred to as swaging. In the flashless die shown in Fig. 133d the volume of the initial slug or blank is equal to or only slightly greater than that of the finished work, and metal flow is restricted in a space confined by the die impression and the punch. This may be called true coining.



Sizing and coining operations are commonly done in *knuckle-joint coining presses*. The general view and gearing diagram of such a press are shown in Fig. 134. Here, crankshaft 1, driven by a motor, reciprocates slide 4 by means of a linkage consisting of pitman 2 and knuckle-joint levers 3. The top half of the die is fastened to the slide. This arrangement enables a high tonnage to be exerted at the bottom of the stroke with a comparatively low torque on the drive shaft. Rated tonnages of coining presses range from 100 to 8000 tons and even more.

Other cold working processes are cold extrusion (impact and forward processes) and cold heading.

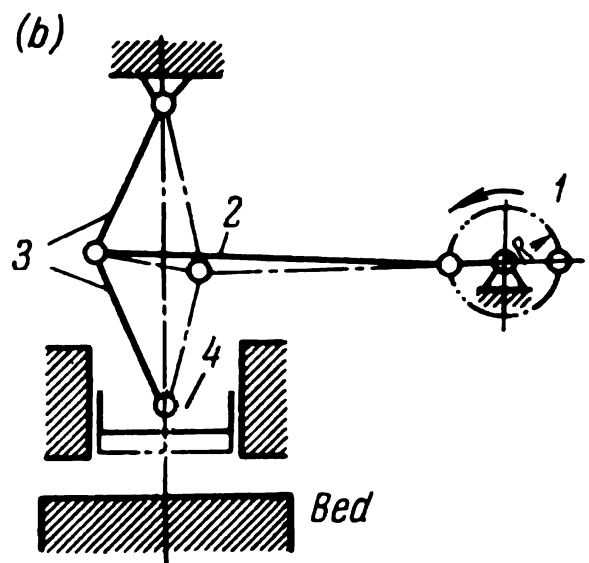
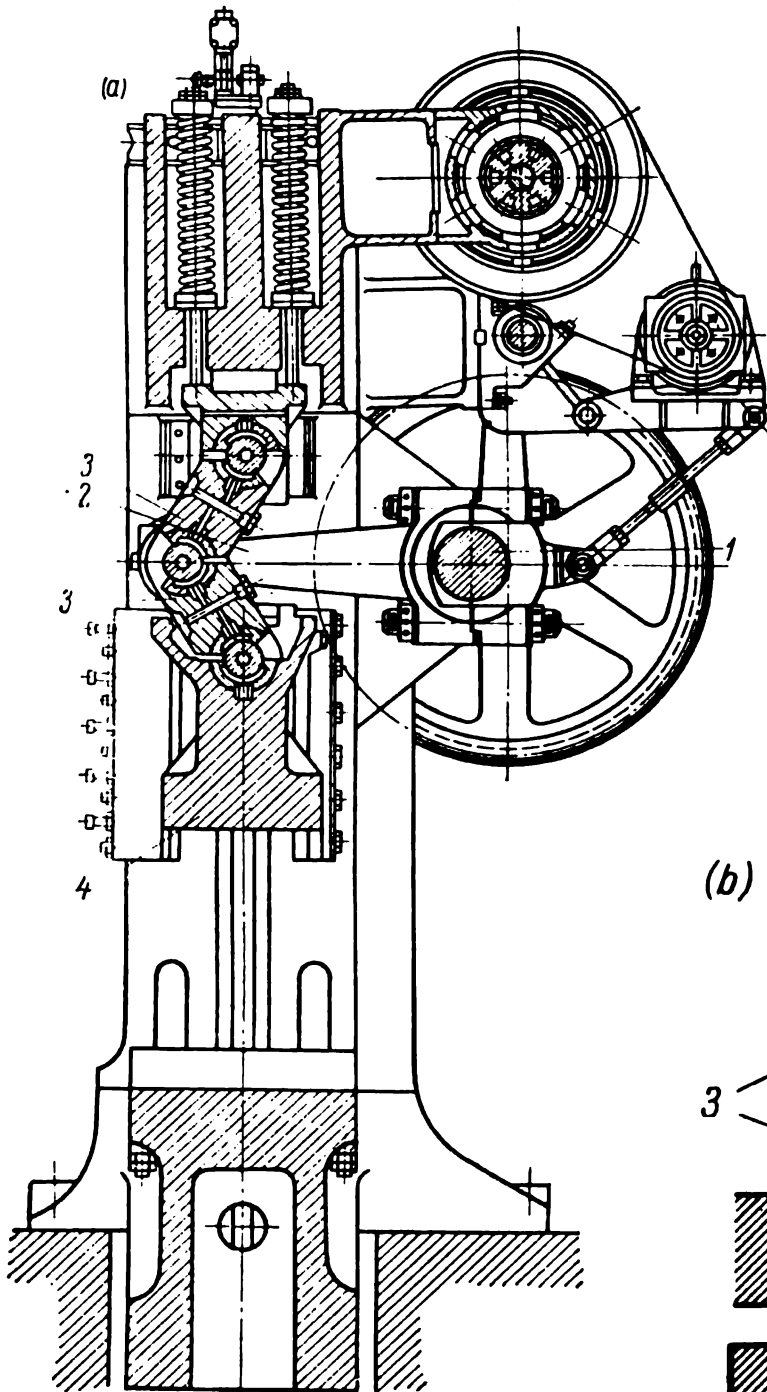


Fig. 134. Knuckle-joint coining press:  
(a) general view; (b) gearing diagram

Cold extrusion is performed in special dies in crank-type and specially designed presses.

*Cold heading* has found widespread application in the mass production of small parts and fastenings (nuts, rivets, nails, etc.) for general engineering, bridge building, construction and other fields

of industry. The stock for these parts is wire or cold-drawn bars from 0.6 to 40 mm in diameter of carbon steel, nonferrous metals and their alloys.

A cold heading operation sequence is illustrated in Fig. 135. The metal bar stock is fed through die 1 to stop 2. Next, the die is shifted to cut off the blank and to carry it over to the heading line where the required head is formed by a blow of punch 3 mounted on the heading gate.

Cold heading in one or several strokes is done chiefly in automatic cold headers in which all operations, including bar (or wire) feed, cut-off, carry-over and ejection of the finished parts are completely automatic. Headers have an output from 20 to 250 pieces per minute.

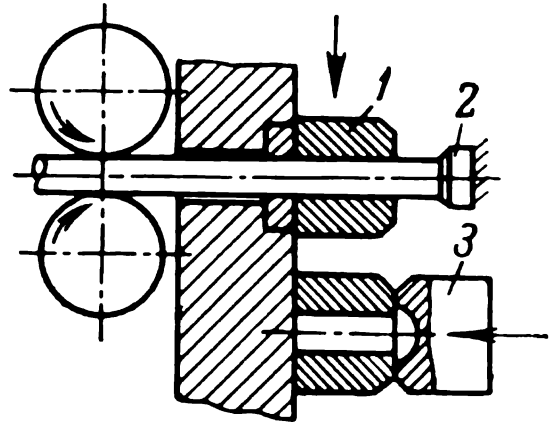


Fig. 135. The cold heading principle

## 21-4. Pressworking of Sheet Metals

Sheet metals are usually pressworked in the cold condition. Pressworking is one of the extensively employed methods of fabricating parts of intricate shape with thin walls.

Since no preheating of the metal is required, the work may be finished to narrow tolerances and with a high surface finish. In the majority of cases pressworked parts do not undergo further machining and are used directly in machine assembly. A disadvantage of cold pressworking is that the metal is less deformable due to its higher resistance and because of work hardening.

The initial material in cold pressworking is band, strip or sheet stock from several hundredths of a millimetre to 5 or 6 mm thick. Stock thicker than 5 or 6 mm is preheated for pressworking.

Sheet metal is pressworked in one or several consecutive operations which can be generally classified as cutting and shaping procedures.

*Cutting operations* include cutting off (shearing), blanking (cutting out along the contour) and others in which the work is severed from the stock.

As a rule, only straight cuts are made in shears and the quality of the cut edge is inferior to that obtained in dies. Therefore, blanks of irregular outline are almost always produced in blanking dies.

In blanking, the stock cut away from the contour of the blank is converted into scrap. This waste of stock can be reduced by proper layout of the blanks on the strip or sheet stock. One such procedure is to increase the number of rows of blanks on the strip, laying out

the blanks in an inclined or staggered arrangement. Commonly, the yield of useful blanks from the stock by weight in cutting operations is at least 70 or 80 per cent.

A blanking die (Fig. 136) consists of the die block, punch, auxiliary parts for locating and guiding the strip, stripper plate, etc.

*Shaping operations* include bending, drawing, forming and others.

In bending (Fig. 137), one or several parts of the sheet blank are bent to specified angles in respect to other parts of the blank.

Drawing is the most complex of cold pressworking operations. Here, a punch forces metal in flat stock to flow between its surface and that of a die so that it assumes a cylindrical, box or cup shape.

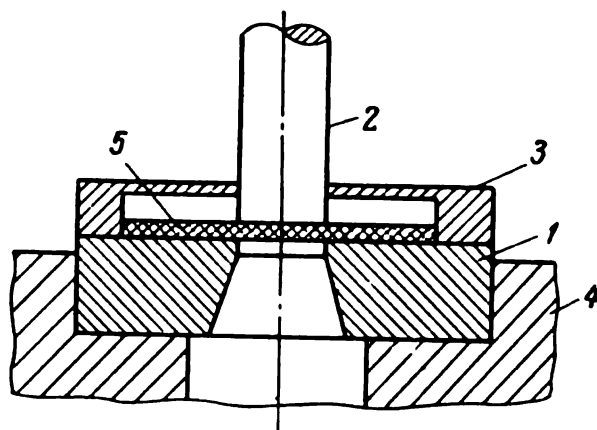


Fig. 136. Principle of the blanking die:

1—die block; 2—punch; 3—stripper plate;  
4—lower shoe; 5—stock

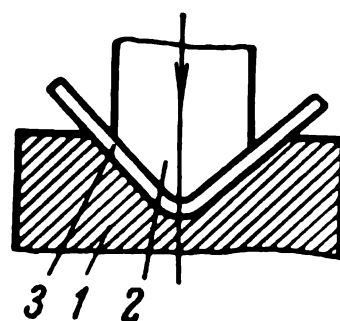


Fig. 137. Bending die:

1—die block; 2—punch;  
3—stock

The operation is done in a draw die (Fig. 138a). The large corner radius  $R$  in the die block 1 enables the flat blank to be smoothly drawn into a cup shape. As the draw punch pushes the flat blank into the die block, wrinkles may be formed in the edges (flange) of the circular blank. Wrinkles may be avoided by using a blankholder 3.

In a single drawing operation a cup may be obtained of a diameter from 1.8 to 2 times less than that of the initial flat blank. Upon more deformation the drawing force required increases to such an extent that the metal is ruptured (the bottom of the cup is torn off). A further reduction in the cup diameter can be achieved only by subsequent drawing (redrawing) operations (Fig. 138b).

There is almost no change in the thickness of the sheet stock in most pressworking operations. An exception is drawing with high degrees of deformation (deep drawing). In this case the thickness of the cup (or shell) near the bottom is reduced (this reduction may amount to 20 or 30 per cent of the blank thickness). In drawing dies the clearance between the punch and die block should be 10 per cent more, on an average, than the stock thickness.

Drawing operations in which the stock thickness is reduced are also employed to some extent. Here, the clearance between the punch and die must be less than the stock thickness.

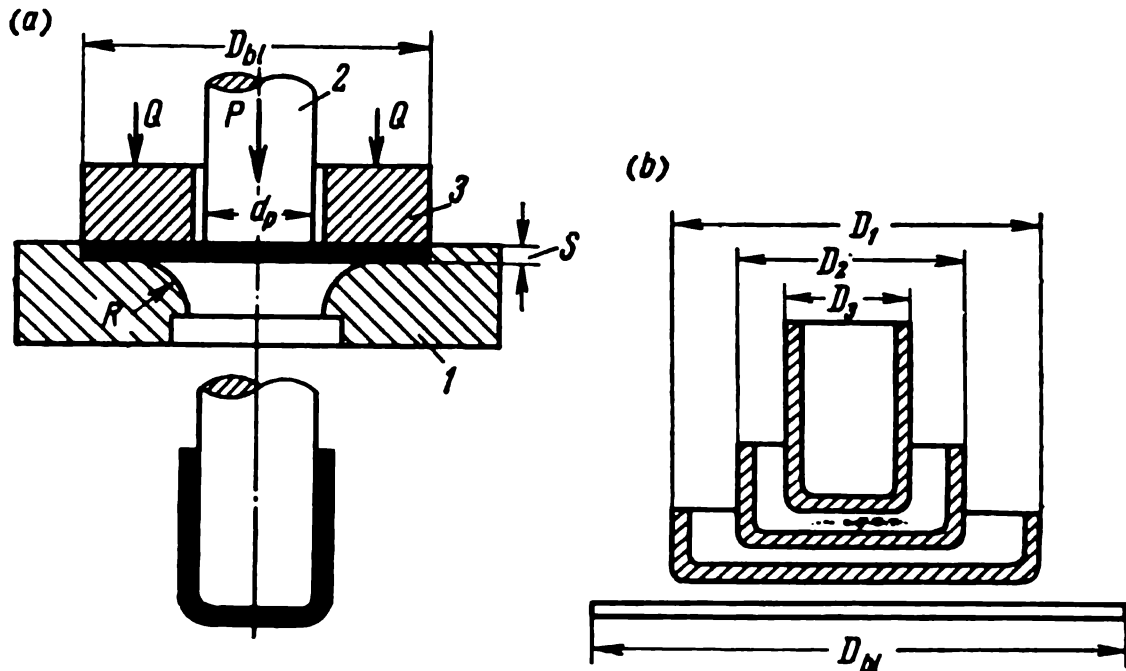


Fig. 138. Draw die for sheet metal

A great variety of dies are employed for drawing operations. The principle of a simple die for the first drawing operation in producing a cup is clear from the schematic illustration of the drawing process shown in Fig. 138a.

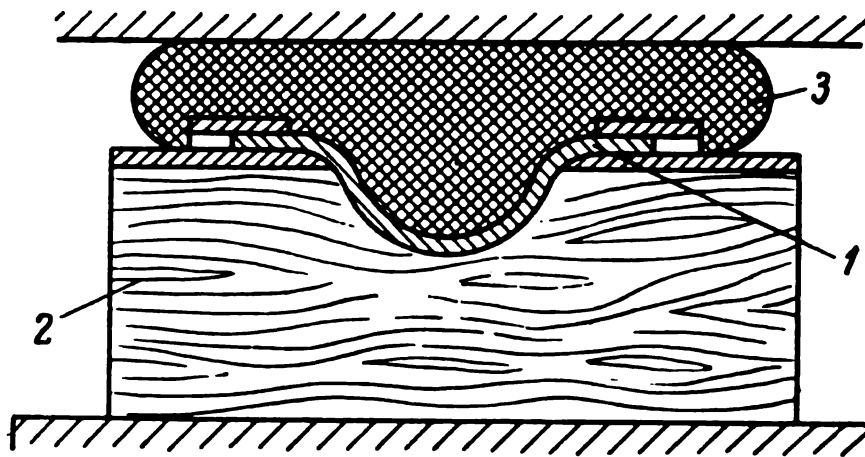


Fig. 139. Rubber-pad processing

A large group of special draw dies are the types used for bulging or expanding the work from inside by means of hydraulic, rubber-pad, compressed air and even explosive processing.

In rubber-pad processing the flat stock 1 (Fig. 139) is placed on a forming block 2 of the required shape. Then the press slide forces

the rubber pad 3 to flow around the forming block so that it shapes the stock to the desired contour. Advantages of this method are the low cost of the tooling, and that many dies may be placed in the press under the slide and compressed simultaneously in one stroke of the press. If a hole with sharp corners is provided in the forming block, work can be punched out or blanked as well as shaped.

The main equipment for pressworking operations consists of crank-type presses of various designs of which the gap-frame and straight-side are the most generally used types.

The external view and gearing diagram of an *open-back inclinable gap-frame crank press* are shown in Fig 140. The drive consists of motor 1,

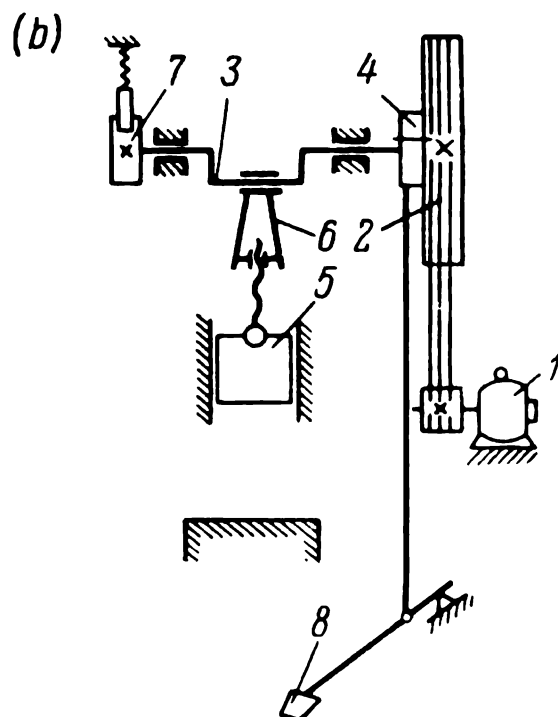
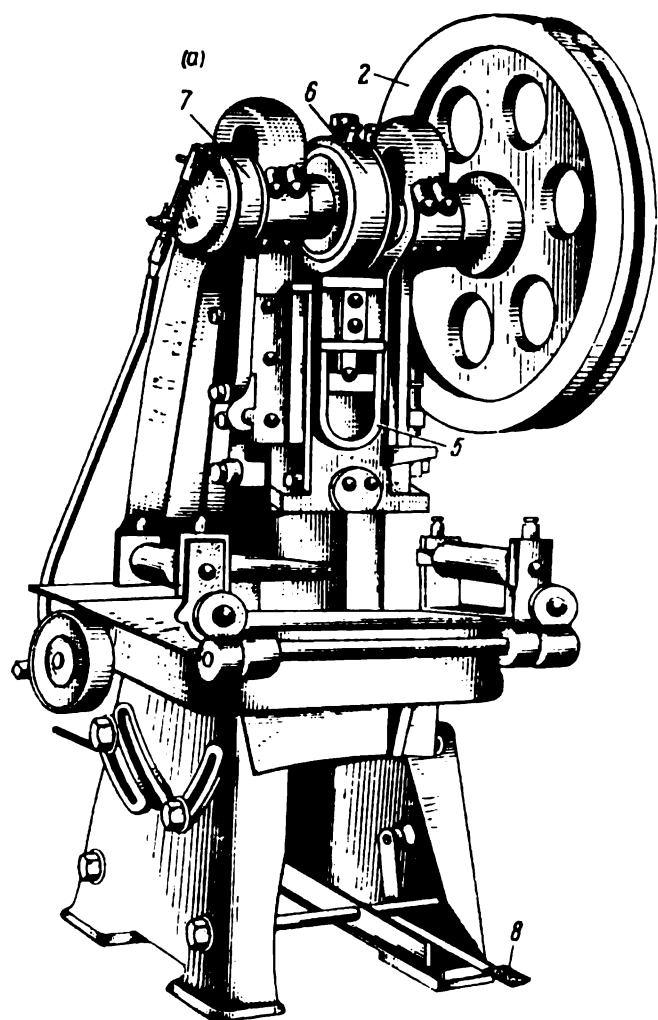


Fig. 140. Open-back inclinable gap-frame crank press:  
(a) external view; (b) gearing diagram

transmitting power through V-belts to flywheel 2 which rotates freely on crankshaft 3. The crank shaft, which is rotated by the flywheel when clutch 4 is engaged, reciprocates slide 5 through pitman 6. Brake 7 stops the crank mechanism and the slide rapidly when the crankshaft and flywheel are disengaged by the clutch. Press operation is controlled by pedal 8 which engages or disengages clutch 4.

Cold pressworking operations have been mechanised and automated to a great extent in Soviet plants. Automatic multiple-operation presses, such as dieing machines, operating at high speeds are widely

used in the mass production of pressworked parts. They perform several operations, such as blanking, bending, drawing, coining, etc., in each stroke of the slide.

*Hot pressworking of plate metal* is employed in the fabrication of heavy work, such as boilers, tanks, ship hulls, etc. The initial material is plate steel over 5 or 6 mm thick, as a rule. The plate is usually cut up into the required blanks mainly by oxygen cutting techniques. The blanks are preheated in batch-type furnaces. Pressworking is done in hydraulic forging or friction screw presses. In some cases, crank-type plate flanging and forming presses are employed for this purpose.

### 21-5. The Mechanical Working of Titanium

Titanium and its alloys are worked at temperatures from 850° to 1000° C. It has been established that titanium is hardened when it is worked at temperatures below 750° C and its brittleness is increased.

Titanium should be annealed before forging operations. Scale should be removed between the rough forging and finishing operations. After forging, the work should be cooled in air and then annealed at 300° C with a holding time of one hour. The rolling temperature of titanium is 100 deg C lower than that required for forging. The rolled stock should be annealed for half an hour at the temperature of about 700° C. Commercial titanium is available in the form of sheets up to 1220 mm wide and 0.3 mm thick, as well as foil.

Titanium can be extruded at a temperature of 950°-1000° C. Contact between the titanium and the die must be avoided as otherwise the metal will stick to and build up on the die, resulting in scoring of the product. Simple methods of preventing contact are oxidation of the surface of the billet and the application of a lubricant consisting of liquid glass and molybdenum disulphide.

Commercial titanium can be subjected to deep drawing operations in the cold state if intermediate annealing is resorted to. The metal is to be annealed at 700°-750° C for 5 to 10 minutes. In the hot condition titanium can be drawn in a single operation.

Pressworking is carried out in the cold state in two operations with an intermediate anneal at 600° C for 15 minutes. Good results are obtained if the titanium is processed within the temperature range from 450° to 550° C and the dies are preheated to 100°-150° C.

Difficulties may arise in wire and tube drawing of titanium due to adherence of the metal to the die and consequent scoring. They may be overcome by producing an oxide film on the metal to prevent contact between the titanium and the drawing die.

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## THE WELDING OF METALS AND ALLIED PROCESSES

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### *Chapter 22*

#### CLASSIFICATION AND CHARACTERISTICS OF THE PRINCIPAL WELDING PROCESSES AND METHODS

##### 22-1. Welding Processes

*Welding* is the process of joining together pieces of metal or metallic parts by bringing them into intimate proximity and heating the places of contact to a state of fusion or plasticity. This leads to interpenetration of the atoms of the metals in the weld zone, and a strong inseparable joint is formed after the metals have cooled.

Welding finds widespread application in almost all branches of industry and construction. Welding is extensively employed in the fabrication and erection of steel structures in industrial construction and civil engineering (frames of industrial buildings, bridges, etc.), vessels of welded-plate construction (steel reservoirs, pipelines, etc.) and concrete reinforcement.

The wide use of welding at the present time is due to its economic advantages over other methods of joining metals, such as riveting, casting, etc.

Welding processes may be classified according to the source of energy employed for heating the metals and the state of the metal at the place being welded. A master chart of the principal welding processes is shown in Fig. 141.

In fusion welding the welding area is heated by a concentrated source of heat to a molten state and filler metal must be added to the weld.

In accordance with the method applied for feeding the filler metal to the weld, welding procedures are classified as manual, semiautomatic or automatic welding.

Pressure welding processes involve the heating of the metallic parts only to a plastic or slightly fused state and forcing them together with external pressure.

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Pressure welding processes are applied to metals which are capable of being brought to a plastic state by heating or due to the action of external forces. It has been established that in this process the most weldable metals prove to be those that have higher thermal conductivity. Such metals more rapidly dissipate heat from the weld

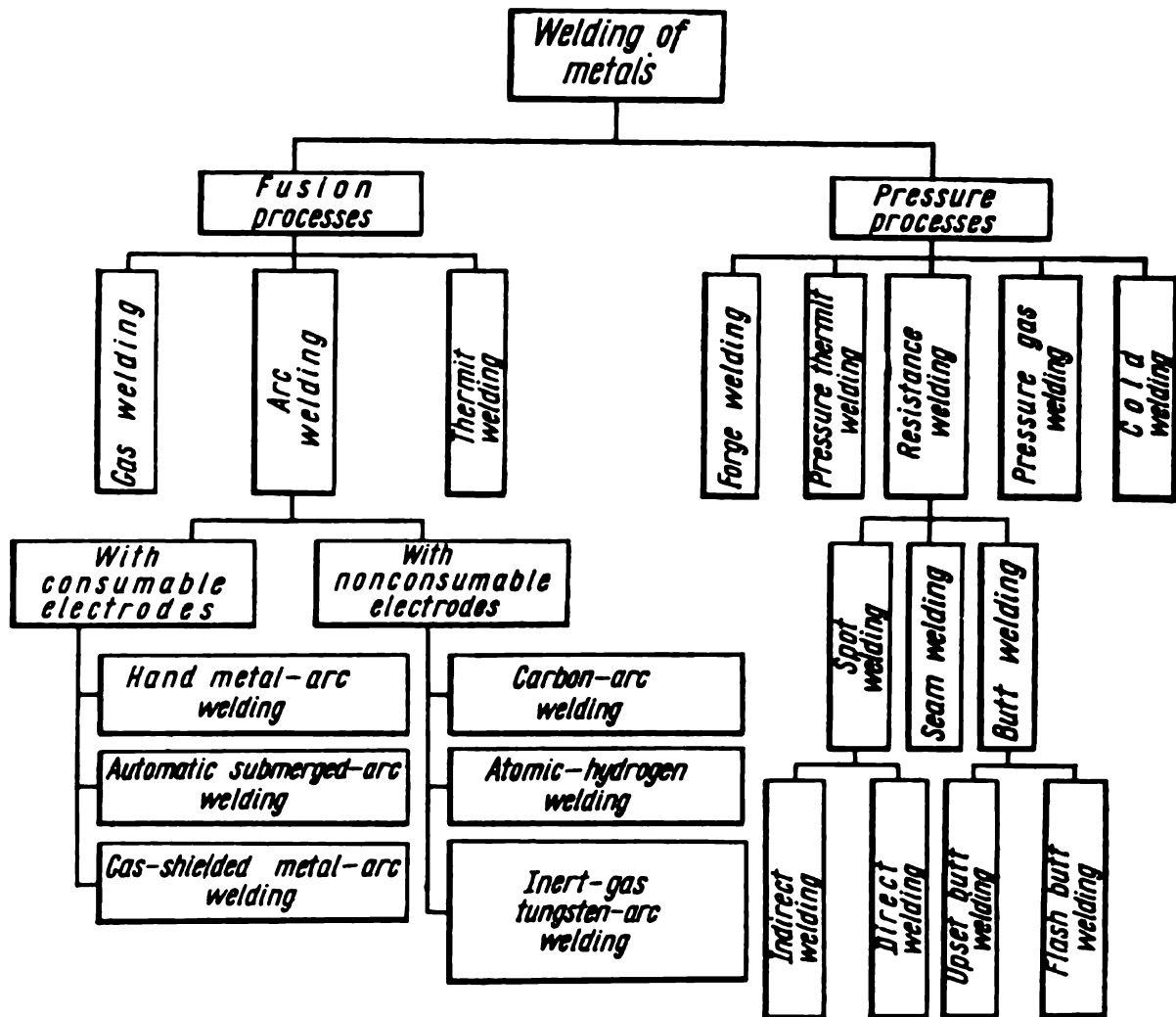


Fig. 141. Master chart of welding processes

zone and do not allow an excessively high temperature to be concentrated in a small area (the latter may lead to considerable internal stress).

The quality of the joint obtained in pressure welding depends to a great extent upon the magnitude of the applied pressure and the temperature to which the metal is heated at the moment of welding. The higher this temperature, the less unit pressure will be required to produce the weld.

Proper cleaning of the surface to be joined is one of the main conditions for obtaining high-quality welds in pressure-welding procedures.



**Types of weld joints.** Welded structures are assembled by butt, lap, corner and tee joints (Fig. 142).

*Butt joints* are formed by welding the end surfaces or edges of the members. The preparation of the edges depends upon the thickness

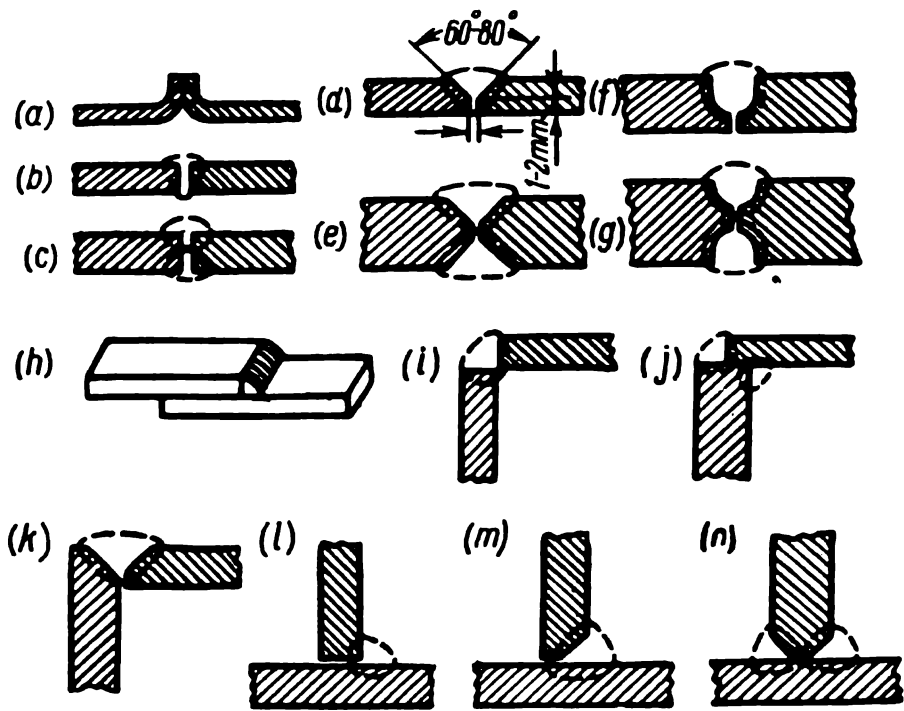


Fig. 142. Types of welded joints

of the metal being welded. Flanging is used for metal up to 3 mm thick (Fig. 142a); the height of the flange should be twice the thickness of the metal. Square butt joints (Fig. 142b and c) with no special edge preparation are suitable for thicknesses of from 3 to 8 mm. Single-V edge preparation (Fig. 142d) is applied for metal from 14

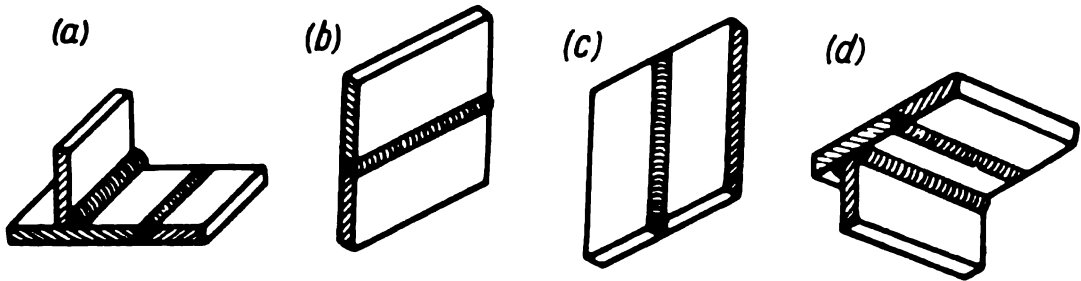


Fig. 143. Positions of welds in space:  
(a) flat; (b) horizontal; (c) vertical; (d) overhead

to 16 mm thick. It is advisable to prepare the edges for a double-V joint (Fig. 142e) if the metal is thicker than 16 mm. U-joints are used for metal over 20 mm in thickness; they may be either single- or double-U types (Fig. 142f and g).

*Lap joints* are produced by fillet welds; the two members being welded should overlap by an amount equaling from three to five

times their thickness (Fig. 142h). Slot, plug and rivet welds are of the lap joint type.

*Corner joints* are welded with or without edge preparation (Fig. 142i, j and k).

*Tee joints* are produced by welding one element to another at an angle of  $90^\circ$  (Fig. 142l, m, and n). Only structures subject to low static loads can be welded without bevelling the edges. Single-bevel joints are employed for critical structures in which the members are from 10 to 20 mm thick and double-bevel designs are used for thicker metal.

As to their positions in space welds may be flat, horizontal, vertical and overhead (Fig. 143a through d).

Also of importance in weld classification is the location of the weld in reference to the direction of the applied stresses. Thus welds may be longitudinal fillet, or side-lap welds, 1, transverse fillet, or end-lap welds, 2, and skew-lap welds 3 (Fig. 144).

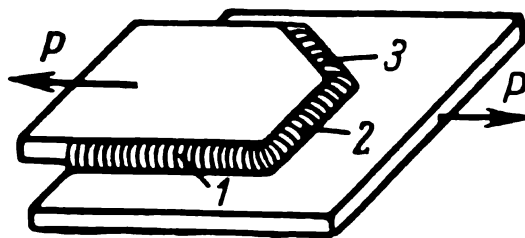


Fig. 144. Weld location in reference to the applied stresses

## 22-2. Metallurgical Processes Occurring in Welding

By far not all metals possess good mutual *weldability*, i. e., the capacity of being welded into reliable inseparable joints having specified properties (definite weld strength, proper structure, etc.).

Metals which are capable of forming a continuous series of solid solutions with each other exhibit the highest degree of weldability. Metals with limited solubility in the solid state have a lower weldability. Metals which are mutually insoluble in the solid state are entirely unweldable by fusion methods. Such metals are heated to a plastic state and then a mechanical force is applied to bring the particles into close contact. In some cases, an intermediate metal is introduced when the metals to be welded are mutually insoluble. This third metal must be capable of diffusing or dissolving in both metals.

Of prime importance in fusion welding are the metallurgical processes occurring in the weld when the weld metal is in the molten state or begins to solidify.

The metal filling and adjoining the weld (near-weld zone) undergoes phase transformations in fusion welding processes due to the rapid heating to the melting point and subsequent cooling.

The melting and cooling conditions in the weld itself and the associated structural changes in the metal of the near-weld zone determine the properties of the welded joint as a whole.

The structure of a single-V weld after solidification and the temperature distribution curve in steel at the moment of welding are shown schematically in Fig. 145. The deposited layer in zone 2 is obtained by the melting of the filler metal and its mixing with the parent metal 1 in the narrow fusion zone 3. In the fusion zone the parent metal is brought to the molten state so that it is rapidly penetrated by the deposited metal and a sound weld is formed.

Various impurities and dirt which get into the deposited metal (oxides, slag inclusions, etc.) are located on the grain boundaries after solidification. This decreases the strength and ductility of the deposited metal. The purer the deposited metal, the higher the mechanical properties of the whole welded joint will be.

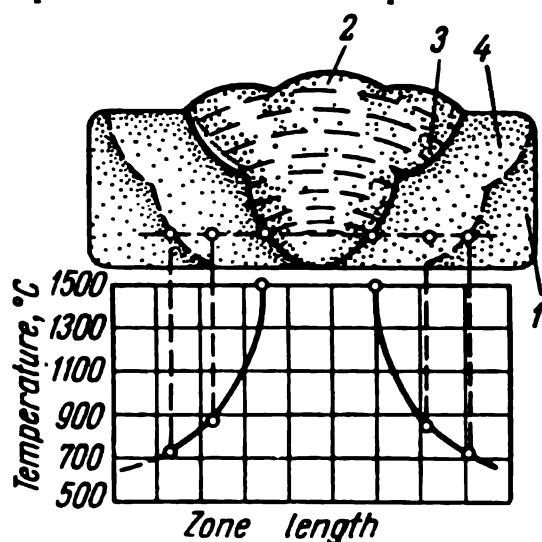


Fig. 145. Weld structure and temperature distribution in welding steel

As a rule, the deposited metal is of columnar (dendritic) structure, characteristic of cast steel. If the deposited metal or the adjacent zone of the parent metal was overheated to a high degree, then in cooling the grains of the parent metal acquire an acicular (needlelike) shape, forming the so-called Widmanstätten structure. The overheated metal has a lower strength and increases the brittleness of the weld.

A zone of heat-affected metal 4 is formed in the parent metal of all welds. In this zone only the structure of the metal is changed by the rapid heating and cooling during the welding process; its chemical composition remains unchanged. The size of the heat-affected zone depends upon the welding method employed and the nature of metals being welded. For example, in hand arc welding with lightly coated electrodes and full-automatic submerged-arc welding the heat-affected zone is of minimum magnitude—2 to 2.5 mm; in welding with covered electrodes it ranges from 4 to 10 mm while in gas welding it may reach 20-25 mm. It is evident that these circumstances affect the structure of the weld.

The weld structure after solidification and the temperature distribution in the metal at the moment of welding low-carbon steel are illustrated in Fig. 146. The diagram shows that next to the metal deposit (section 0-1 on the temperature curve) is the fusion zone (section 1-2) in which a transition from the structure of the deposited metal to that of the parent metal is observed. Partial melting of the metal occurred in this zone and for a certain time it was a mixture of the solid and liquid phases.

The weld structure after solidification and the temperature distribution in the metal at the moment of welding low-carbon steel are illustrated in Fig. 146. The diagram shows that next to the metal deposit (section 0-1 on the temperature curve) is the fusion zone (section 1-2) in which a transition from the structure of the deposited metal to that of the parent metal is observed. Partial melting of the metal occurred in this zone and for a certain time it was a mixture of the solid and liquid phases.

Next to the fusion zone is section 2-3 where the metal was overheated to a high degree which led to grain coarsening and the formation of an acicular (Widmanstätten) structure. This is the most brittle part of the weld and the weak point of the welded joint.

The temperature of the metal in section 3-4 did not exceed  $1100^{\circ}\text{C}$ . Here normalisation of the steel is observed, accompanied by the formation of a fine-grained structure since the temperature to which

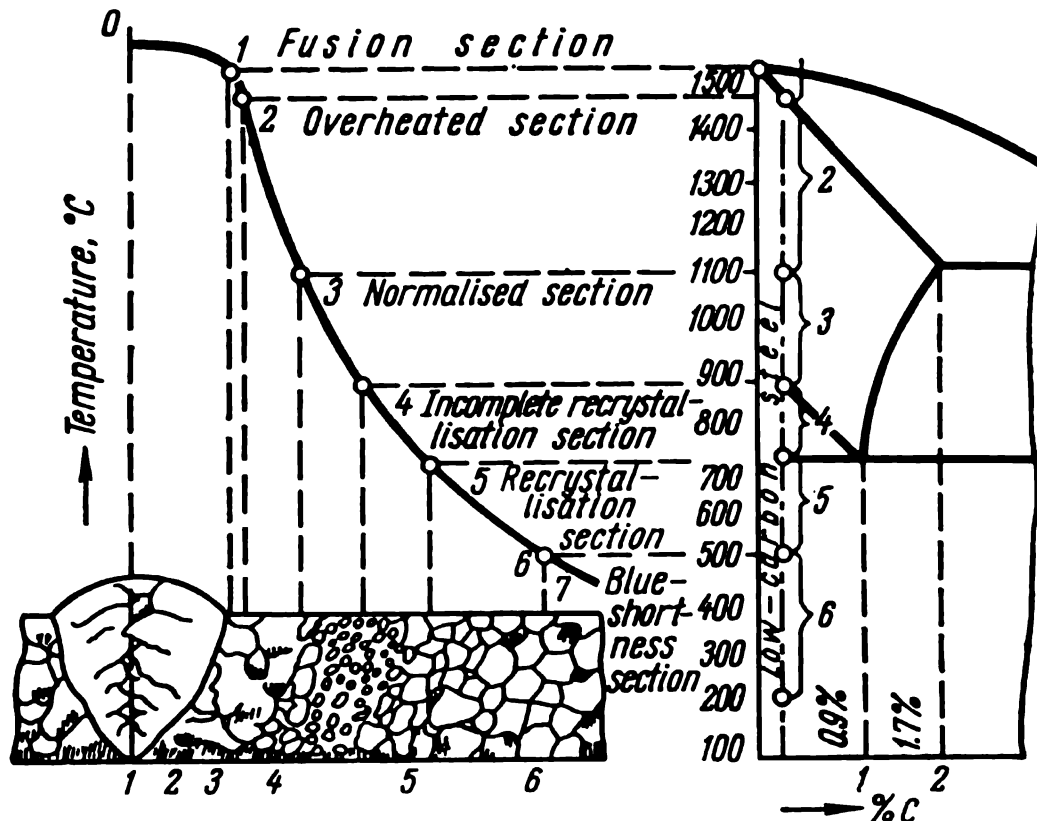


Fig. 146. Structural changes in the heat-affected zone when welding steel

the metal was momentarily heated was only slightly above critical point  $A_c$ . The metal in this section has improved mechanical properties in comparison to the metal of the unaffected zone.

Incomplete recrystallisation of the metal occurred in section 4-5 since it was heated to a temperature between the critical points  $A_c$  and  $A_{c_1}$ . In addition to coarse grains of ferrite, fine grains of ferrite and pearlite are formed in this part of the weld.

No structural changes occur in the metal of section 5-6 if it was not subjected to plastic deformation prior to welding. However, if the metal was worked before welding, then recrystallisation is observed in this section, i. e., the previous form and size of the grains, broken up and altered in plastic deformation, will be restored.

The structure of the metal in section 6-7, called the blue-shortness section, does not differ from that of the parent metal.

The structural changes of the parent metal in the heat-affected zone of the weld discussed above have a negligible effect on the mechanical properties of low-carbon steel regardless of the welding method being employed. In welding certain grades of structural steel, however, hardened structures may be formed in the heat-affected zone of the weld which sharply reduce the ductility of the welded joint and frequently lead to crack formation.

## *Chapter 23*

### ARC WELDING

#### **23-1. Properties of the Electric Arc**

Arc welding is the most extensively employed method of joining the components of metallic parts; here the source of heat is an electric arc.

An electric arc is a continuous stream of electrons flowing through some sort of medium between two electrodes and accompanied by intense heat generation and radiation.

To strike an arc it is necessary to ionise the air gap or specially provided gaseous medium between the electrodes. This requires a certain amount of energy which is determined by the ionisation potential needed to release the electrons from their bonds with the atomic nuclei.

Various gases have different ionisation potentials. The lower the ionisation potential the easier the given gas is ionised and the more stable the arc will be though its temperature will be lower.

The air or gas in the arc gap is continuously ionised by electrons emitted by the surface of the negative electrode (cathode). These electrons collide with the molecules of the gaseous or vaporous substances in the space between the electrodes and split them into simpler components—positively charged ions and electrons. Two factors influence the emission of electrons from the cathode: (1) high temperature (thermionic emission) and (2) the strength of the electric field (autoelectronic emission).

The negatively charged ions (electrons) bombard the anode; positively charged ions—the cathode. Mutual bombardment of the ions with a sufficiently high arc voltage transforms kinetic energy of these particles into thermal and luminous energy.

Thermal and luminous energy is not uniformly evolved in the welding arc. About 43 per cent of the total amount of heat is evolved on

the anode and about 36 per cent on the cathode. The remaining 21 per cent is evolved by the arc.

The temperature of an electric arc depends upon the type of electrodes between which it is struck. It will be about  $3200^{\circ}\text{C}$  on the cathode and about  $3900^{\circ}\text{C}$  on the anode for carbon electrodes and  $2400^{\circ}\text{C}$  and  $2600^{\circ}\text{C}$ , respectively, for metal electrodes. In the centre of the arc the temperature reaches  $6000^{\circ}\text{C}$ - $7000^{\circ}\text{C}$ .

Only from 60 to 70 per cent of the heat is utilised in arc welding to heat up and melt the metal. The remaining 40 to 30 per cent is dissipated into the surrounding space.

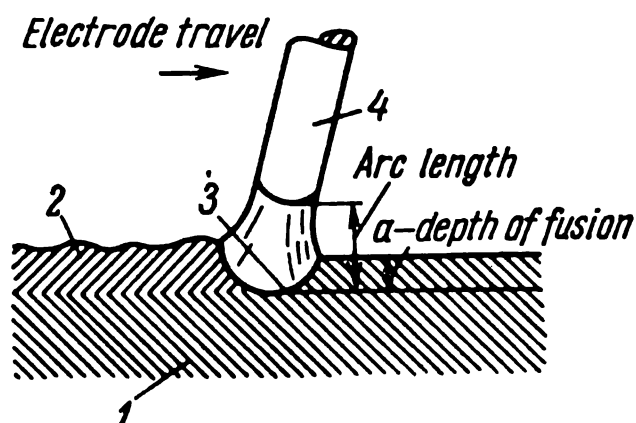


Fig. 147. An electrical welding arc:  
1—base metal; 2—deposited metal; 3—crater;  
4—metal electrode

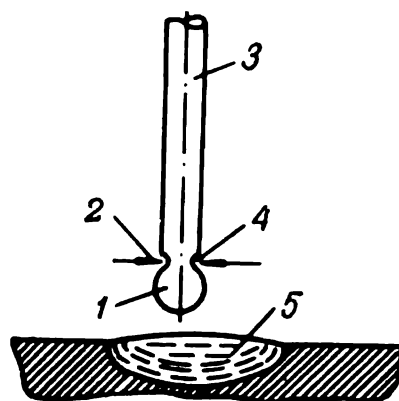


Fig. 148. Formation of a drop of molten metal:  
1—drop; 2—electromagnetic forces promoting necking; 3—metal electrode; 4—neck; 5—pool

A welding arc, struck in passing a current between a metal electrode (rod) and the parent metal (body of the workpiece), is shown schematically in Fig. 147. Here it is evident that the arc discharge (arc flame) has the shape of a column widening towards the surface of the part. At the foot of the column within the part, the arc crater, or welding pool, is formed. To strike the arc the electrode is shorted on the work so that the current begins to flow. Due to the high ohmic resistance at the point of contact the end of the electrode and the adjacent area of the work are intensely heated. This facilitates the forming of the arc discharge at the moment the electrode is retracted from the work surface. The radiant and thermal energy generated by the arc melt the end of the electrode and the area of the work under it to form the welding pool into which drops of molten metal flow from the electrode.

The gap between the molten end of the electrode and the surface of the pool is occupied by an incandescent medium which is a mixture of partially ionised air and the gaseous and vaporous substances that appear at high temperatures in the interaction between the electrode material and its chemical coating and the air.

The stable arc required for high-quality welding can be achieved with an arc length equal from 0.6 to 0.8 of the electrode diameter. The arc length is defined as the distance between the end of the electrode and the surface of the molten metal on the work. Commonly, up to 90 per cent of the total metal of a consumable electrode flows as drops from the electrode to the welding pool. The other 10 per cent does not reach the pool because of spattering, vaporisation and oxidation. The drops of molten metal are carried over from the electrode to the pool by the forces of gravity, surface tension, pressure of the gases evolved in the metal and the constructive action of an electric current on the molten metal (pinch effect). Surface tension imparts a spherical shape to the drops and brings their size to the optimal value before they break off. This facilitates the flow of drops to the pool (Fig. 148). Surface tension holds the pool on the weld, this circumstance being obviously of prime importance in overhead welding operations.

The pressure of the gases generated when the electrode melts also helps to carry over the drops to the work.

The electric current contributes in a peculiar manner to molten metal transfer from the electrode to the pool. When a drop of metal begins to break away from the electrode, a neck is formed which is rapidly reduced in cross section by the constricting (pinch) effect of the current. This leads to an increase in the current density in the neck, promoting its rupture and imparting an impulse to the drop in the direction of the pool.

Only a comparatively low potential difference is required between the electrodes to strike an arc. From 40 to 45 V is usually sufficient for direct current and from 50 to 60 V for alternating current. The voltage falls after the arc is established. A stable arc can be maintained between a metal electrode and the work metal with a voltage of from 15 to 30 V while from 30 to 35 V is needed if the arc is between a carbon or graphite electrode and the metal. The voltage necessary for proper arc maintenance depends upon the composition of the electrode rod, its coating, the gas pressure in the surrounding medium and the magnitude and type of current, but the main factor is the length of the arc.

Arc stability is much higher with direct current than with alternating current. The reason for this is that the arc gap cools when an alternating current arc is periodically extinguished and re-established. The stability of the arc can be increased by raising the voltage or by increasing the frequency of the alternating current.

### **23-2. Basic Types of Arc Welding**

On the basis of the type of electrode employed arc welding can be classified into: (1) processes using a nonconsumable electrode (carbon or tungsten) and (2) processes using a consumable metal electrode.

Welding is done with a carbon electrode according to the diagram illustrated in Fig. 149*a* in the first method which was developed by N. Bernardos. Direct current is employed and the electrode is connected to the negative pole and the work to the positive pole of the generator. A very unstable arc is obtained if welding is carried out with reversed polarity (plus on the nonconsumable electrode and minus on the work).

The second method of welding was invented in 1890 by the Russian engineer N. Slavianoff (Fig. 149*b*); it is widely applied in the engineering industries, building construction and for various types of repairs.

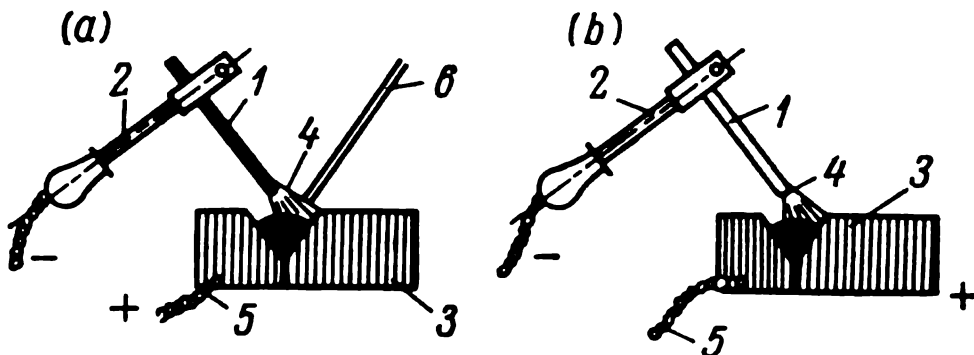


Fig. 149. Arc welding processes:

(a) Bernardos process; (b) Slavianoff process; 1—electrode; 2—holder; 3—work being welded; 4—electric arc; 5—flexible wire; 6—filler metal

In the welding of metals the arc may be either direct or independent. A direct arc is struck between the electrode and the work. An independent (indirect) arc is established between two carbon electrodes adjacent to the parts being welded. The metal is heated by the indirect action of the arc.

The surrounding medium has a harmful effect on the quality of the weld being made. Various methods have been devised to protect the weld.

The three types of welding arcs are submerged, shielded and open.

The most effective protection of the metal against the action of the surrounding medium is afforded by an arc submerged in water or under a blanket of granular fluxes.

Of the widest practical application are methods in which the metal is protected by a layer of slag or gaseous envelope. The slag is formed around the arc by the use of coated electrodes on which the surface covering melts when heated. Sometimes an envelope of gases is provided to enclose the welding zone and to isolate the arc from the surrounding medium. Simple noncritical welding operations are most frequently performed with a bare electrode (open arc).

Both direct and alternating currents are employed in arc welding.



The advantages of direct-current welding lie in the higher arc stability and the possibility of using direct and reversed polarity to regulate the degree to which the work is heated.

Alternating-current welding, however, is more extensively used in general practice because of its many favourable features. Equipment for a-c welding is considerably less expensive, lighter in weight, smaller in size and simpler in operation due to the absence of moving parts. The efficiency of a-c welding transformers varies from 0.8 to 0.85 while d-c outfits have an efficiency from 0.3 to 0.6.

The electric energy consumption per kilogram of deposited metal in a-c welding is from 3 to 4 kWh while for d-c welding it is as high as 6 to 10 kWh.

One disadvantage of a-c welding is the comparatively low power factor ( $\cos \varphi$ ) at the welding station, usually not exceeding 0.3 or 0.4. The motor in a d-c welding outfit has a power factor of 0.6 to 0.7.

### 23-3. Electrodes for Arc Welding

Nonconsumable and consumable electrodes are used for arc welding.

*Nonconsumable electrodes* may be of the carbon, graphite or tungsten types. The carbon and graphite varieties are used only in d-c arc welding. Tungsten electrodes are used for both a-c and d-c welding procedures in the atomic-hydrogen and inert-gas shielded-arc processes.

*Consumable electrodes* may be made of various metals (steel, cast iron, copper, brass, bronze, aluminium or cemented carbides) depending upon their purpose and the chemical composition of the metals to be welded. Both bare and coated electrodes are widely employed.

Consumable electrodes for hand arc welding are made in the shape of rods (bare or coated) up to 12 mm in diameter and 450 mm long. Soft steel wire containing from 0.1 to 0.18% C and 0.025 to 0.04% P and S is used for electrodes in welding carbon steels. Electrodes for alloy steels are made of low-alloy steel wire containing up to 0.25% C.

As a rule, bare electrodes are used for automatic welding procedures.

*Coated electrodes* are employed in many welding processes and can be divided into two general groups: (1) lightly coated electrodes with a coating layer several tenths of a millimetre thick (the weight of the coating is from 1 to 5 per cent of the electrode weight) and (2) covered electrodes with a relatively thick high-quality covering applied in a layer of from 1 to 3 mm. The weight of such a covering is from 15 to 30 per cent of the electrode rod.

The primary purpose of a light coating is to increase arc stability and for this reason they are also called ionising coatings. This type

of coating contains chalk (80 to 85 parts by weight) and water glass (15 to 20 parts by weight).

Lightly coated electrodes are used in welding only noncritical structures. The welds produced have poorer mechanical properties due to the lack of protection of the molten metal.

The heavy coating on covered electrodes improves arc stability, produces a protective layer of gases and slag surrounding the arc and drops of molten metal to prevent oxidation and nitrogen contamination and also retards cooling of the welding pool. If necessary, alloying elements may be added to the coating which pass into the weld metal, improving its mechanical properties. Heavy coatings are composed of ionising (chalk), slag-forming (kaolin), gas-generating (starch), deoxidising (aluminium, ferromanganese, etc.), alloying and binding materials.

### 23-4. Arc Welding Machinery

In direct-current welding the arc is supplied from welding generators; in alternating-current welding—from welding transformers.

Arc welders may be classified as *single-operator* types for supplying one arc and *multiple-station* types for supplying several arcs. Arc welders may be either *stationary* or *portable* in design; they may be powered by an electric motor or an internal combustion engine.

Sources of power for supplying a welding arc must have external characteristics differing from those used for ordinary purposes such as lighting, motor supply, etc.

The external characteristic of power sources for ordinary consumers is illustrated by curve 2 in Fig. 150. This curve, indicating the voltage  $E$ , approximates a straight line parallel to the axis of abscissas along which the current, in amperes, is plotted. This form of the external characteristic is chosen because the power consumers—incandescent lamps and electric motors—require a constant voltage in the supply circuit for normal operation without regard for changes in the load.

Entirely different requirements are made to the power source of a welding arc. In this case a constant voltage characteristic would lead to a continuously increasing arc and the welding current would increase without restriction until the leads were destroyed by a short circuit.

A falling, or drooping, characteristic is required of the power source to avoid the aforesaid and to maintain a stable arc. Here the voltage on the terminals of the welder decreases with an increase in the load, and vice versa (curve 1 in Fig. 150).

Any changes in the arc length result in corresponding changes in the voltage and current according to curve 1. The steeper the falling

characteristic, the less the variations in current will be upon changes in arc length.

Arc welding processes demand maximum possible constancy of the rated welding current; the arc length must have the least feasible effect on the magnitude of the current.

In a single-operator welding generator with split poles, a line diagram of which is shown in Fig. 151, poles of the same polarity are arranged adjacently (in this case two north and two south poles) to produce two perpendicular magnetic fluxes; one flux  $F_s$  between

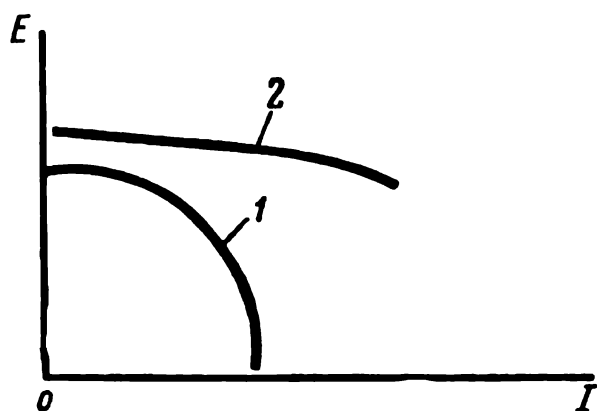


Fig. 150. External characteristic of the power source

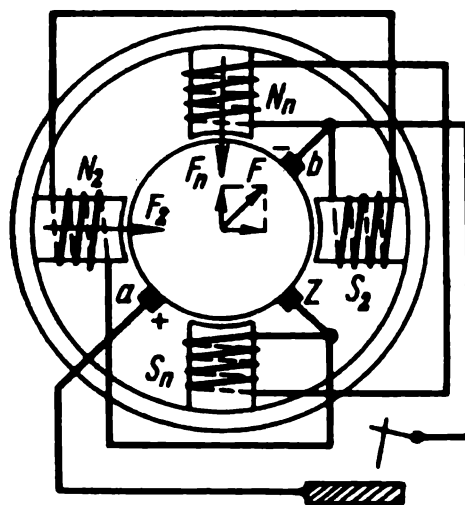


Fig. 151. Circuit diagram of a welding generator with split poles

$N_s$  and  $S_s$  and the other  $F_n$  between  $N_n$  and  $S_n$ . In construction, this generator resembles a four-pole type but since poles of the same polarity are adjacent it is, in effect, a two-pole generator.

The generator has two field windings (one constant and the other variable) and three brushes on the commutator—two main brushes  $a$  and  $b$  and an auxiliary brush  $z$  which supplies the field winding. Both field windings are connected to the main  $b$  and auxiliary  $z$  brushes.

The voltage between the main and auxiliary brushes remains constant; practically it does not depend on the load and is equal to approximately 24 V. This constancy of the voltage over these brushes is due to the fact that they take voltage off the wires of the armature that are under the longitudinal poles at the given moment. The longitudinal poles are always in a state of high magnetic saturation and the magnetic flux in them is maintained almost constant notwithstanding considerable variations in the magnetomotive force.

The current is controlled by shifting a brush along the commutator and by means of a rheostat in the field winding.

The type CYT-2p welding set, manufactured in the U.S.S.R., operates as described above. It consists of a d-c welding generator

of the described design and a three-phase asynchronous motor linked together by a flexible coupling. The generator has a rating of 7.25 kW, current of 280 A and a working voltage of 30 V. The set is designed for supplying one arc. It is mounted on a truck for transportation to the site where the welding operation is to be done.

When welding operations are to be done at new building sites or under field conditions where no electric energy is available, use is made of portable welding sets comprising a d-c welding generator and an internal combustion engine. The generator and engine are mounted on a common frame and linked together with a flexible coupling. The

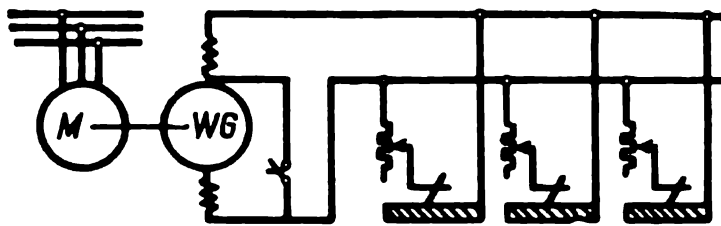


Fig. 152. Welding stations supplied by a multiple-operator generator

type CAK-2 portable set is ordinarily mounted on an automobile or trailer. The generator operates with a rated power of 7.5 kW, current of 250 A and voltage of 30 V.

Large welding departments employ a multiple-operator system of power supply. Up to nine stations are connected to one supply unit. The supply circuit is illustrated in Fig. 152.

The Soviet multiple-operator set, type ПСМ-1000, consists of a motor and a generator rated at 60 V and 1000 A. The current is regulated at each station separately by means of special rheostats called ballast resistors.

In a-c welding the arc is supplied from a welding transformer. An inductive resistance, called a current regulator or reactor, is connected in series with the supply circuit for regulating the welding current and improving arc stability. The current regulator also enables the drooping external characteristic to be obtained.

Alternating-current welding apparatus of a great many types is being manufactured in the U.S.S.R. at the present time.

Type CTЭ transformers with separate current regulators (Fig. 153) are of the single-phase step-down type and are connected to 220, 380 or 500 V power mains (the secondary voltage is equal to 55 and 65 V).

The current regulator by means of which the current is controlled in the transformer consists of the split core and the reactance coil that produce the inductive resistance. At the short-circuit current, the arc voltage drops to zero and the reactance coil provides a steep

drooping external characteristic. The inductive resistance and, consequently, the welding current are changed by varying the air gap in the regulator core. When the gap is increased the magnetic resistance of the magnetic circuit in the coil is likewise increased, and

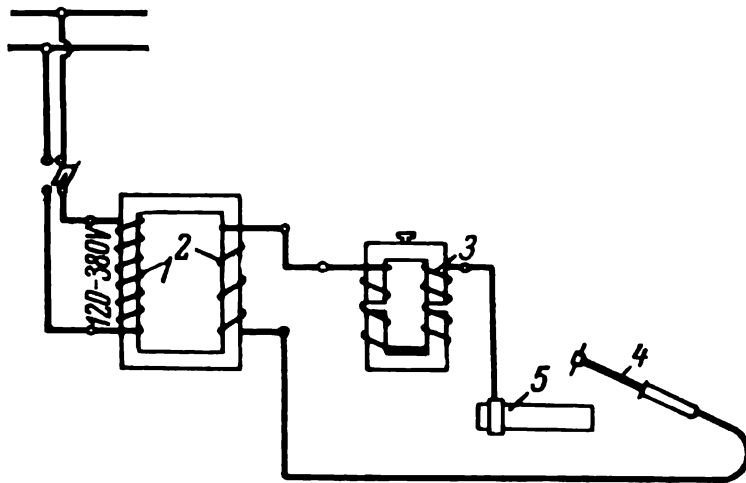


Fig. 153. Circuit diagram of the CTЭ welding transformer:

1—primary winding; 2—secondary winding; 3—current regulator; 4—electrode holder; 5—work being welded

the magnetic flux in the coil is reduced. This leads to a reduction in the self-induced current flowing in a direction opposite to the welding current. The welding current increases with a reduction in the self-induced current and decreases when the air gap in the core is reduced.

The type CTЭ welding transformers are designed to give a current from 60 to 700 A.

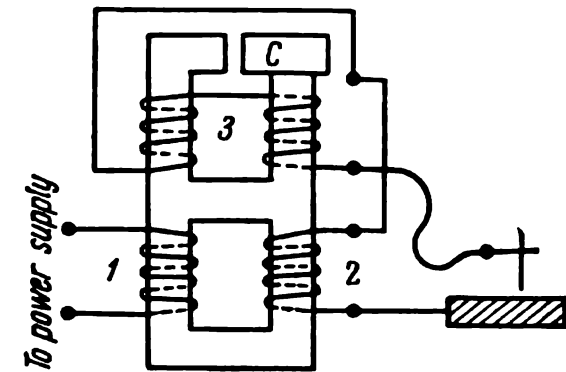


Fig. 154. Principle of the type CTH transformer

Transformers, type CTH (Fig. 154), with built-in current regulators, designed by Academician V. Nikitin, consist of a common magnetic circuit with three coils: primary 1, secondary 2 and reactance 3.

The combined action of coils 1 and 2 produce the main magnetic flux. The magnetic flux produced by reactance coil 3 is directed opposite to the main flux and therefore, in welding, the arc voltage is the difference in voltages of the secondary and reactance coils.

The welding current is regulated by shifting the laminated pile C built up of sheet iron.

CTH transformers are employed for currents of 500, 1000 and 2000 A in both hand and automatic welding.

Transformers with built-in regulators are used only for single-operator welders.

Multiple-operator welding systems commonly use three-phase transformers with a secondary voltage of 65 to 70 V when the secondary coil is star-connected. In this case each welding station is furnished with its own current regulator.

The power rating of the transformer is based on the total power requirements of all the stations, taking into account, as well, the simultaneity factor of their operation.

Type CTЭ transformers may also be used for multiple-station operation if the welding currents are relatively low and only 2 or 3 arcs are to be supplied.

### 23-5. Hand Arc Welding

The main factor in high-quality welding is a proper selection of the welding conditions which are determined by the electrode diameter, welding current and the productivity.

The diameter of the electrode is selected to suit the thickness of the metal being welded and the type of joint (Table 4).

Table 4

Electrode Diameter Selection According to the Thickness  
of the Metal to be Welded

Metal thickness, mm . . . . .	up to 2	2 to 4	4 to 6	6 to 8	over 8
Electrode diameter, mm . . . . .	2	3	4	5	5-6

The magnitude of the welding current depends upon the thickness of the welded metal, type of joint, welding speed, position of the weld, the thickness and type of the coating on the electrode and its working length but, for practical purposes, it is ordinarily determined by the selected electrode diameter.

The welding current, in amperes, can be found from the formula:

$$I_a = kd, \quad (84)$$

where  $k$  = coefficient of proportionality expressed in A/mm; hand welding  $k=45$  to 60 A/mm for ordinary steel electrodes, 18 to 22 A/mm for graphite and 5 to 8 A/mm for carbon electrodes;

$d$  = diameter of the electrode, in mm.

In general practice welding is performed with currents over 50 A.

The arc voltage depends only upon the arc length and is determined from the formula

$$E_a = \alpha + \beta l, \quad (85)$$

where  $\alpha$  and  $\beta$  = constant factors;  $\alpha = 10$  to  $12$  and  $\beta = 2$  to  $3$ ;  $l$  = arc length, in mm.

**Metal-arc welding.** An arc is established by bringing the electrode in contact with the work at the point at which the weld is to be started after connecting the work to the welding circuit. After a light contact the electrode is immediately withdrawn to a distance of from 2 to 4 mm from the work. Then the welder feeds the electrode uniformly to the work, maintaining a short arc.

*Direction of welding*

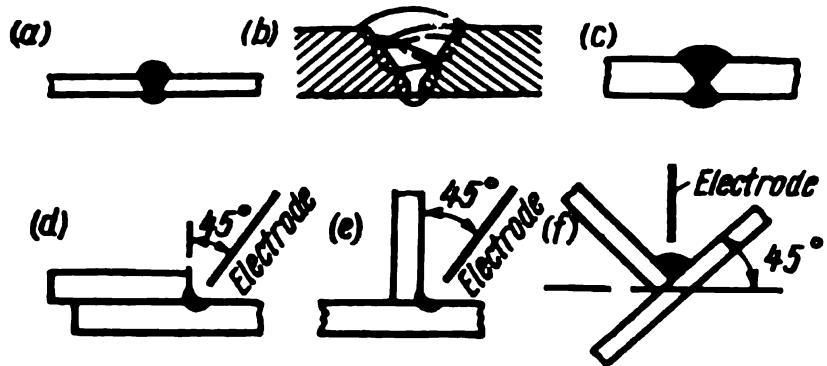
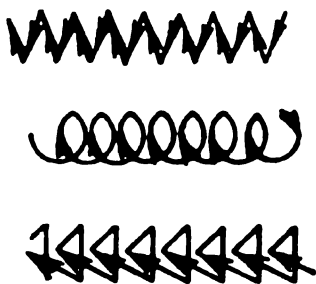


Fig. 155. Path of the electrode in hand arc welding

Fig. 156. Depositing the weld bead for butt and corner joints

The electrode is held at an angle of from  $15^\circ$  to  $20^\circ$  from the vertical on a flat weld to produce a weld bead. By changing the angle of inclination of the electrode the welder can vary the depth to which the parent metal is melted and the cooling rate of the welding pool.

In welding thin plates or sheets a narrow bead is deposited (with a width from 0.8 to 1.5 electrode diameters) without transverse oscillation. Spreading beads are also used. Three movements are accomplished by the tip of the electrode in such welds: (1) feed along the axis of the electrode, (2) progressively along the weld and (3) an oscillating or circular movement across the weld. Transverse oscillating movements improve the heating of the joint edges, retard cooling of the molten pool of deposited metal, enable a more homogeneous weld to be obtained and eliminate inadequate joint penetration. Diagrams of the oscillating motions of the electrode tip are shown in Fig. 155.

A square butt joint (Fig. 156a) is welded by depositing a spreading bead on one side of the joint, melting the edges of the metal through the full thickness. Sometimes, when the position of the weld allows, after chipping out the overlap and slag, a back weld with a narrow bead is made on the opposite side.

When a single-V joint is welded the arc is struck near the root of the joint and a bead is deposited (Fig. 156b). Depending upon the

thickness of the metal and the diameter of the electrode the weld is filled with metal in one or several passes; reinforcement is provided on the weld. In multilayer welding each layer is carefully cleaned before depositing the next layer.

Double-V joints are welded by alternating the beads, welding on one side and then on the other, to keep the joint symmetrical and to reduce warpage in welding (Fig. 156c).

In welding a corner joint the electrode is held at an angle of  $45^\circ$  to the work surface (Fig. 156d and e). When higher current values are used the two parts to be welded are inclined to an angle of  $45^\circ$  with the horizontal (in the flat position with the face of the weld approximately horizontal) to avoid poor fusion (Fig. 156f).

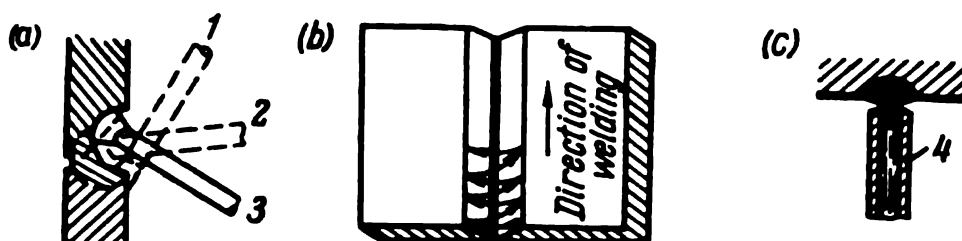


Fig. 157. Procedures in making various types of welds:  
1, 2 and 3—positions of the electrode; 4—electrode coating

Only the upper plate is bevelled for a weld in the horizontal position; the arc is struck on the lower edge and is moved gradually over to the bevelled upper edge (Fig. 157a).

It is difficult to perform welding in the vertical position because the molten metal flows downward. This can be reduced by welding with a short arc and running the weld upwards (Fig. 157b). This procedure cannot be applied for sheets up to 1.5 mm in thickness.

Welds in the overhead position are made with a very short arc and using electrodes with a refractory coating which forms a cup around the electrode in operation thus holding the molten filling metal (Fig. 157c).

In metal-arc welding the thermal action of the arc melts the end of the electrode and the edges of the work. It has been established that the drops of molten metal pass along the arc at a velocity of 40 m per second and that the number of drops varies from 10 to 30 per second.

**Carbon-arc welding.** The arc is struck in carbon-arc welding by the Bernardos method in the same manner as for metal-arc procedures.

The carbon electrode is very slowly consumed. The arc remains stable; an increase in the arc length up to 6-10 mm has no appreciable effect on the quality of the weld.



Carbon-arc welds on sheets up to 3 mm thick with flanged edges (Fig. 158a) are performed without filler metal which, however, is fed into the arc when welding thicker sheets or plates (Fig. 158b).

In addition to direct arcs, arcs with independent action may be employed. In the latter case, two carbon electrodes are clamped in a special holder.

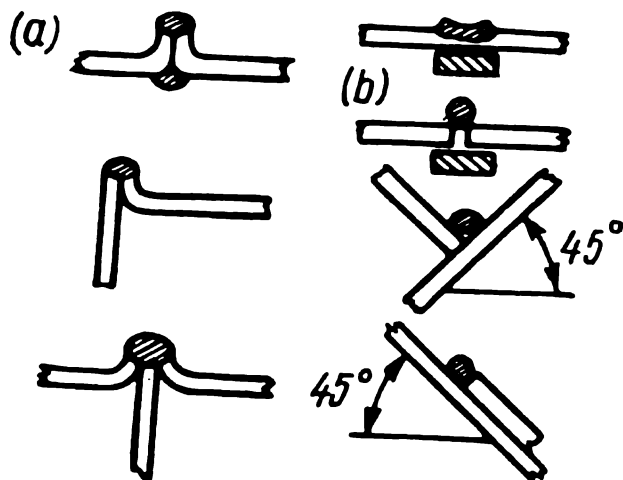


Fig. 158. Types of carbon-arc welds

Carbon-arc welding procedures ensure ample strength of the joint and a weld of satisfactory appearance.

The rate of welding with a carbon electrode and without filler metal reaches 50 to 60 m per hour for sheet 1-3 mm thick. The diameter of carbon electrodes varies from 10 to 25 mm for current values from 200 to 600 A.

Carbon electrodes are used principally for welding nonferrous metals, such as copper, aluminium, zinc and their alloys, as well as for surfacing with cemented carbides.

### 23-6. Highly Efficient Hand Arc Welding Procedures

The rate of hand arc welding can be substantially increased by applying new, progressive techniques.

One such method is *deep penetration* hand welding which raises the labour productivity of welders by 150 to 200 per cent. This method involves the use of electrodes with a coating having a higher melting point than the electrode metal.

After establishing the arc the covered electrode is held tightly against the work. During welding, a small hood of unmelted covering is formed on the end of the electrode; the arc is inside this hood. The edge of the hood prevents the electrode from short-circuiting on the work thereby enabling the heat of the arc to be more efficiently utilised and more complete fusion to be obtained.

Another high-speed procedure is welding with a *bundle of electrodes* (Fig. 159). Here welding is done simultaneously with two, three or more electrodes insulated from one another. This allows a higher current density per square millimetre of electrode cross section and, consequently, the welding speed can be increased by 30 or 40 per cent.

The arc is first struck on the electrode nearest to the workpiece since the ohmic resistance is lowest over the shortest gap. As soon as this electrode melts down to some extent, the distance between it and the work increases and an arc is struck on the one of the other

electrodes that happens to be closer to the work than the first electrode. Further deposition of metal from the bundle of electrodes is continued in the same manner, i. e., the welding arc is transferred periodically from one electrode to another.

Welding with a *three-phase arc* is still another high-speed method. In comparison with single-phase welding it increases the labour productivity by 2 or 3 times, saves up to 25 per cent electric energy and ensures a deeper fusion on the work. Three-phase welding (Fig. 160) consists in the following: each of the two phases of the power source 1 is separately connected to two

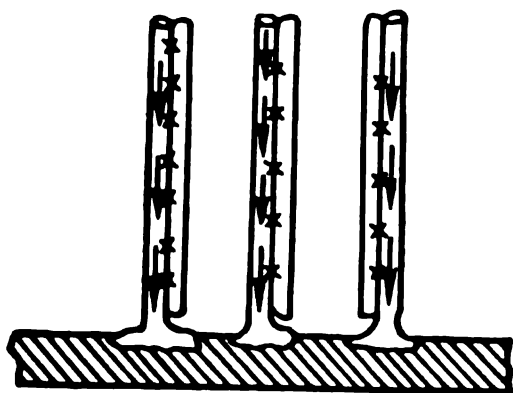


Fig. 159. Welding with a bundle of electrodes

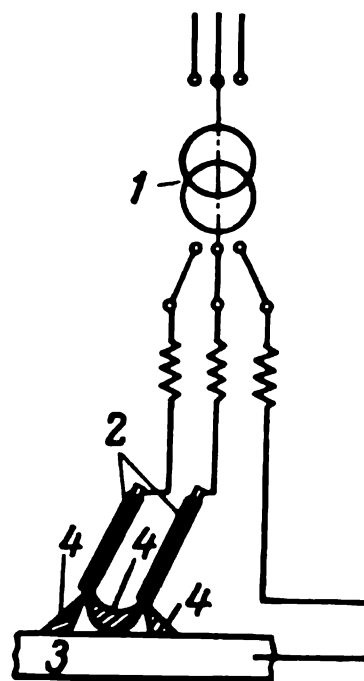


Fig. 160. Welding with a three-phase arc

parallel covered electrodes 2 and the third phase is connected to the work 3. Thus, when the circuit is closed three arcs 4 are established: one between each of the electrodes and the work and one between the electrodes themselves. This arrangement not only increases the rate of welding but also ensures high stability of the process since if one of the arcs fails the other two will continue to operate.

### 23-7. Automatic Arc Welding

In automatic arc welding the arc and the feeding of filler material to the weld are controlled by special mechanisms. Designs of automatic welders were first proposed by the inventors of arc welding, N. Bernardos and N. Slavianoff. The idea of automatic submerged-arc welding is also theirs. In 1923 submerged-arc welding was developed to the practical stage by D. Dulchevsky.

**Automatic submerged-arc welding.** A new automatic submerged-arc welding method, in which a heavy layer or blanket of flux is applied on the place prepared for welding, was worked out in 1940

in Electric Welding Institute of the Ukrainian Academy of Sciences under the supervision of Academician Y. Paton.

The application of a submerged arc affords many advantages over other processes. The power of the arc can be considerably increased (up to 20-150 kW and even more), the welding rate is increased by 5 to 10 times in comparison with hand welding, and the deposited metal has a higher purity and better properties than in hand arc welding.

Automatic submerged-arc welding is done with an automatic welder (welding head) which feeds the bare electrode wire into the welding zone and automatically maintains an arc of definite length.

The equipment required and the process of submerged-arc welding are illustrated in Fig. 161.

The automatic welder incorporates the following units into a single assembly: (a) welding head 3 which feeds the wire into the arc zone, (b) reel 4 on which the coil of electrode wire is put, (c) hopper 6 for flux with hoses for feeding the flux and for picking up excess flux by suction and (d) self-propelling drive 7 by means of which the whole welder is traversed along the weld at a given speed (welding speed).

The work 1 with prepared edges is arranged under the welder on racks or it is held by special manipulators. Current from the power supply (usually a-c since such power sources are simpler in design and less expensive) flows to the work and to the welding head. In operation, flux from the hopper is fed through a hose to the prepared joint in advance of the electrode. The arc is established between the work and the electrode wire 2. The arc forms the molten metal welding pool which is completely submerged in the molten slag and the flux 5 that remains unmelted. Since the arc is under a layer of flux it is not visible; no protection is required for the operator's eyes nor shielding for the other personnel. The unmelted flux is picked up by suction hose 8 and returned to hopper 6. The weld 10 has smooth, fine-rippled surface of silver colour; it is covered by a layer of fused slag 9 which pops off by itself or is easily removed.

Current values ranging from 3000 to 4000 A are commonly used in this process. This leads to considerable concentration of heat in the welding zone and a large pool of liquid metal and molten flux is formed. These features call for edge preparation and welding procedure differing from other methods.

The formation of a submerged welding pool is shown schematically in Fig. 162. The arc melts not only the electrode and the base metal but a part of the flux as well. The molten metal of the electrode in the form of separate drops mixes with the molten flux and settles to the bottom of the pool. The vaporous and gaseous matter evolving at the high temperature of the arc are the products of partial vaporisation of the metal and of the decomposition of the fluxes and air remaining in the layer of granulated flux; these products form an

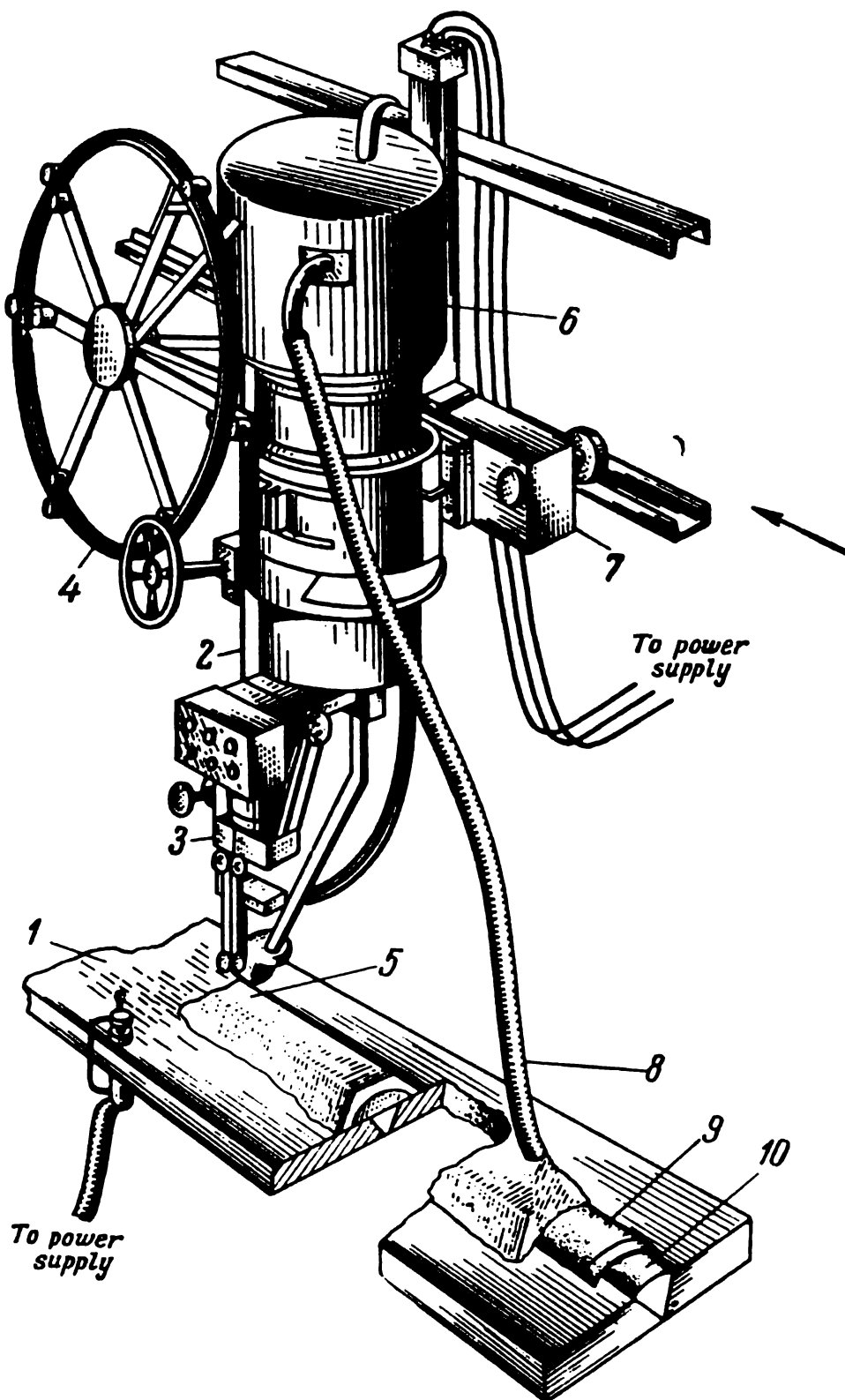


Fig. 161. The automatic submerged-arc welding process

enclosed gas cavity (gas bubble) in the vicinity of the arc. This cavity is closed from above by a layer of molten flux (slag) which isolates the arc from the ambient atmosphere.

A certain overpressure developed in the arc cavity by the thermal expansion of the arc gases promotes the formation of the weld. The overpressure in conjunction with the pressure of the gases pushes the products of welding back in the direction opposite to that in which the weld is being run. Only a thin layer of the base metal remains at the foot of the arc stream where the crater is formed. A powerful

welding arc ensures deep penetration and fusion of the base metal under such conditions.

If the thin slag shell over the gas bubble is accidentally fractured and the gases are released so as to allow the liquid slag to settle in the pool and short the arc, then the fresh lot of vapours and gases that are formed will push the slag upwards and restore the gas cavity required for normal arc operation.

The flux melted by the electrode mixes in the arc stream with the drops of metal and intensively floods them. This promotes a rapid development of the reaction between the metal and the slag; the reaction is also favoured by

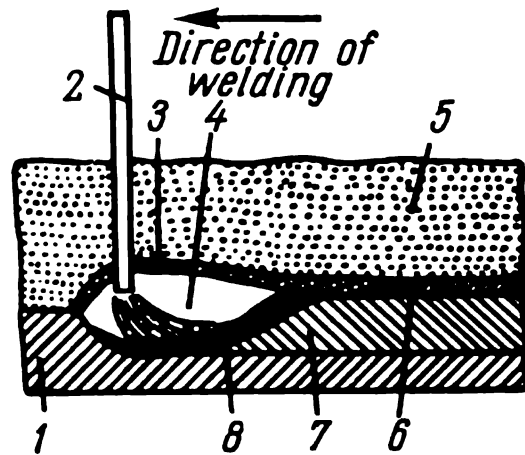


Fig. 162. Formation of the submerged-arc welding pool:

1—base metal; 2—electrode wire; 3—molten flux; 4—gas bubble; 5—flux; 6—solidified slag; 7—deposited metal; 8—liquid metal

the high temperature in the arc gap. As the metal solidifies the slag floats to the surface and also solidifies forming a shell above the metal pool.

In the course of electrode and automatic head travel along the weld and as the weld cools, the metal and flux vapours crystallise in the direction from the external layers to the core to form the deposited metal.

The liquid slag has a lower melting point than the metal and, therefore, solidifies later thereby retarding the cooling of the weld metal.

The fact that the metal remains in the liquid state for some time in automatic welding enables the pool to be more thoroughly freed of nonmetallic particles, slag inclusions and gases so that the metal acquires a more homogeneous chemical composition.

In automatic welding the properties of the weld are determined, in the main, by the composition of the materials employed, i. e., the compositions of the metal being welded, electrode wire and flux. The surface of the electrode wire must be free of rust and dirt. These

conditions are met by cold-drawn wire, grades CB-08, CB-08A, CB-15, CB-OAG, CB-08GA and CB-15Г.

The processes of oxidation of manganese and other components of the deposited metal are of special importance in welding low-carbon and structural steels. For this reason the flux should be selected of a composition that will properly deoxidise the metal in the pool and that contains manganese and other additives to make up for the elements burned out in welding. The melting point of fluxes used in automatic welding should not exceed  $1200^{\circ}\text{C}$ ; their viscosity in the molten state should be low. Such are the frequently applied Soviet grades АИ-3, АИ-10 and ФЦ-6 containing ferrosilicon, manganoous ore, limestone, dolomite, feldspar and alumina.

Ceramic fluxes for automatic welding have been developed by K. Khrenov from the components of ordinary electrode coating compounds for hand welding. Ceramic fluxes are mixed on a water glass base with the addition of the necessary alloying elements thereby providing the possibility of varying the chemical composition of the deposited metal.

The Paton Electric Welding Institute has applied a new type of hollow electrode wire for use in alloying the deposited metal when automatic submerged-arc welding is done with ordinary fluxes. The hollow wire is filled with a powder consisting of various ferroalloys.

**Submerged slag-pool welding.** The electrical slag-pool welding method was also developed by the Paton Electric Welding Institute of the Ukrainian Academy of Sciences. Characteristic features of this method are the controlled weld size and penetration and the arcless welding process. The process is shown schematically in Fig. 163.

The components to be welded are set in a vertical position with a distance of from 15 to 30 mm between the butted edges depending upon their thickness. The welding wire 1 and the flux are fed automatically into the clearance. Several electrode wires can be fed simultaneously if the parts to be welded are thick enough. Water-cooled copper slides 2 travel upwards along the joint. They prevent the slag and molten metal from spilling out of the pool, cool the pool of metal and form the weld 6. An arc is established between the steel backing plate 3 and the electrode wire only at the very beginning

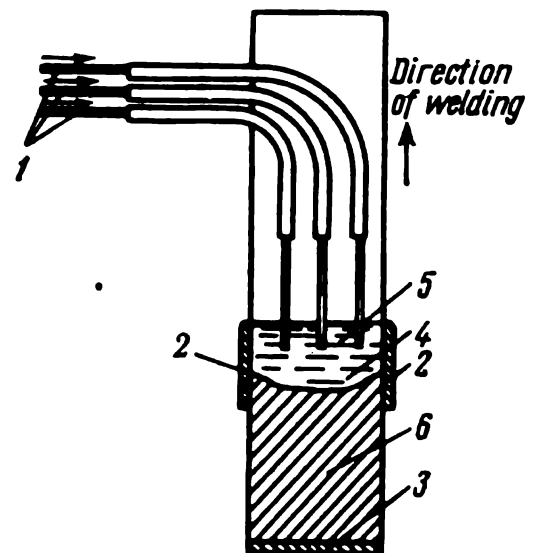


Fig. 163. Principle of the automatic submerged slag-pool welding process

of operation before a pool of molten metal 4 and flux 5 of sufficient volume has accumulated. Further melting of the filler and base metal is due to the current flow between the electrode, pool and slag. The molten base metal and that from the electrode sink to the bottom of the slag pool where they form the weld in the joint between the components.

The chief advantage of electrical slag-pool welding is that very thick components (up to 400 mm) can be welded together in one pass without bevelling the edges. A very clean weld is obtained, free from slag inclusions.

This method is predominantly applied in heavy engineering in the manufacture of beds and frames for heavy machinery, drums, boilers, etc.

### 23-8. Gas-shielded Arc Welding

The principle of gas-shielded arc welding consists in the introduction of a gas around the arc to shield the arc and the metal against contact with the ambient air. The inert gases, argon and helium, are used for shielding, as well as hydrogen, carbon dioxide and others.

The most widely used of the gas-shielded procedures are argon-arc, atomic-hydrogen and carbon-dioxide-shielded arc welding.

**Argon-arc welding** (Fig. 164). Argon-arc welding may be done either with a nonconsumable tungsten electrode or with consumable electrodes.

In inert-gas tungsten-arc welding the arc is shielded by an atmosphere of argon or helium. The filler rod is fed to the arc zone. The arc may be supplied by either direct or alternating current; both hand and automatic procedures are applied. The holder (torch) of the tungsten electrode (Fig. 165) also serves to provide a flow of inert gas around the electrode and arc and to transfer the welding current to

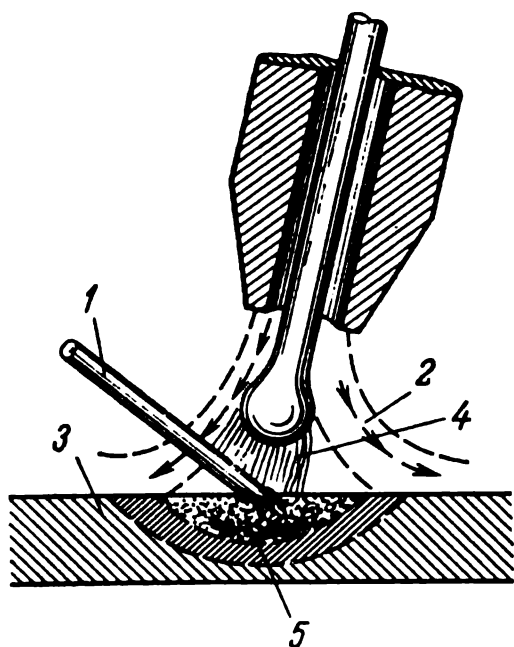


Fig. 164. The argon arc welding process:

1—filler rod; 2—flow of shielding gases; 3—work being welded; 4—arc; 5—deposited metal

the weld zone. Welding is done from right to left.

Wire, grade CB-10FC, is used in welding low-carbon steel with a consumable electrode by the argon-arc method. This wire has an increased manganese and silicon content for better deoxidation of the weld, suppressing boiling of the molten metal and eliminating electrical couples in the deposited metal.

Argon-arc welding is used chiefly for joining stainless steels, aluminium, magnesium and titanium alloys, as well as for critical steel parts up to 3 or 4 mm in thickness. The argon consumption ranges from 400 to 1200 litres per hour (from 600 to 800 litres per hour for medium welding conditions) while the voltage is from 10 to 12 V for nonconsumable tungsten electrodes.

**Atomic-hydrogen welding.** This process differs from the preceding in that an alternating-current arc is formed between two tungsten or carbon electrodes along which streams of hydrogen are fed to the welding zone (Fig. 166). The molecules of hydrogen are dissociated by

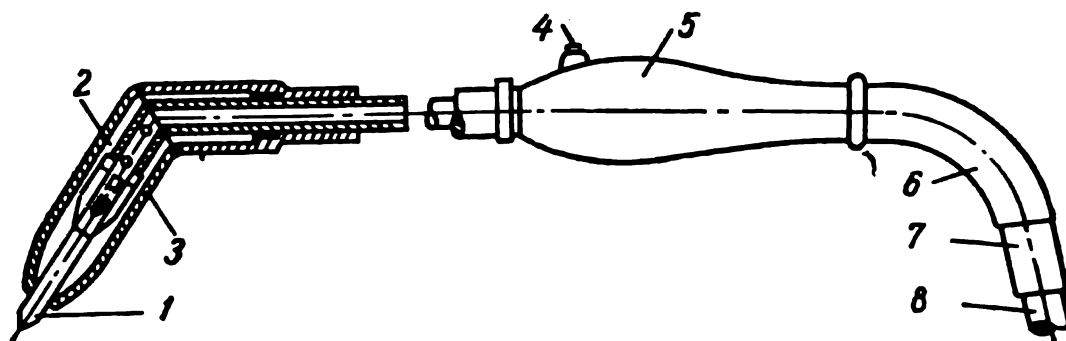


Fig. 165. Torch for argon-arc welding:  
1—tungsten electrode; 2—gas passages; 3—clamp; 4—insulated tube; 5—handle;  
6—insulation; 7—current conductor; 8—gas supply

the high heat of the arc in the gap between the electrodes. The formation of atomic hydrogen proceeds with the absorption of heat:  $\text{H}_2 \rightleftharpoons 2\text{H} - 100,600 \text{ cal per mol}$ . The stream of atomic hydrogen strikes the cooler surfaces of the metal and is again converted into molecular hydrogen.

The metal is heated by the indirect arc and by heat evolved when the atomic hydrogen is converted into the molecular form according to the reaction  $2\text{H} \rightleftharpoons \text{H}_2 + 100,600 \text{ cal per mol}$ . This produces a flame with a temperature of about  $3700^\circ\text{C}$  which shields the welding pool from oxidation and from saturation with nitrogen.

The transformers employed for atomic-hydrogen welding have an open-circuit voltage of up to 300 V and an operating voltage up to 120 V; in accordance with the thickness of the metal welded the welding current ranges from 20 to 100 A while the hydrogen consumption varies from 350 to 800 litres per hour.

Atomic-hydrogen welding is applied in joining steels and light alloys, when the components are from 2 to 10 mm thick, in welding critical designs operating under dynamic loads, high pressure, or in a vacuum as well as in other cases when the weld must be airtight and possess high mechanical properties.

**Carbon-dioxide-shielded arc welding.** The arc in this method is established between a bare wire (consumable electrode) and the work.



The arc is shielded by an atmosphere of carbon dioxide which is fed to the welding zone through the external nozzle of a special torch similar to that used in argon-arc welding.

A schematic diagram of a unit for semiautomatic carbon-dioxide-shielded arc welding is shown in Fig. 167.

The deposition factor (g per hr per ampere) in this process is higher than in submerged-arc welding. With straight polarity this rate is from 1.5 to 1.8 times higher than with reverse polarity. This process

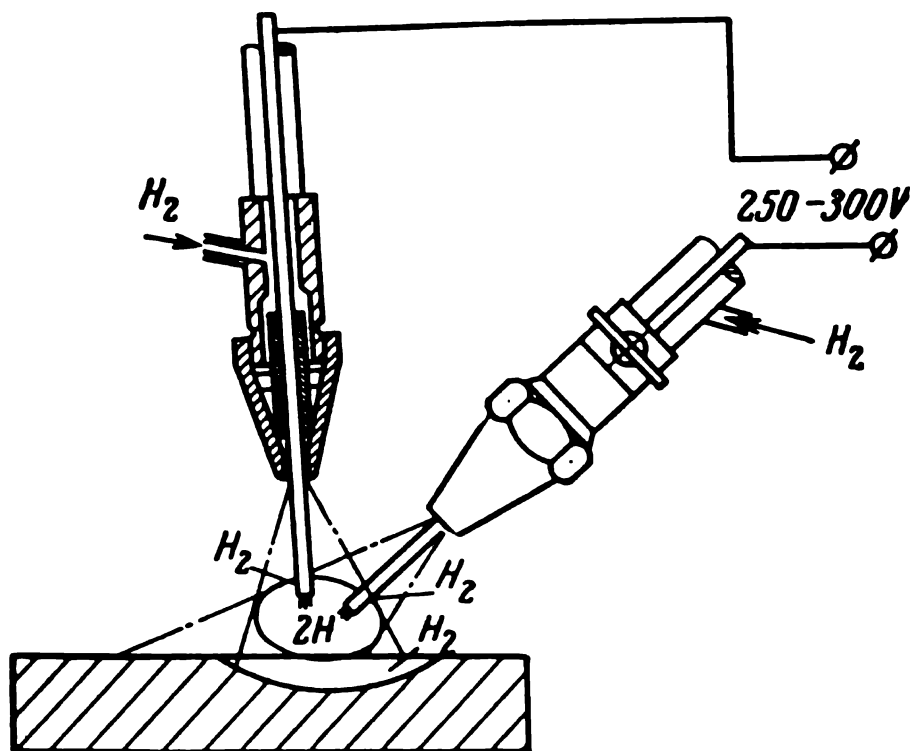


Fig. 166. Principle of the atomic-hydrogen welding process

is distinguished for its high rate of production which may reach 18 kg per hour of deposited metal. The welding rate reaches 60 m per hour. The productivity is from 2.5 to 4 times higher than for hand welding and 1.5 times higher than in submerged-arc welding. The carbon dioxide used for this process must be dry and have a purity not lower than 98 per cent and, for critical structures, not lower than 99 per cent (gas from the cylinder is passed through a dryer to eliminate moisture).

Wire, grades Cв-08ГСА and Cв-0852CA, from 1.6 to 2.5 mm in diameter is employed for electrodes in carbon-dioxide-shielded arc welding.

A current value from 250 to 500 A and an arc voltage from 26 to 34 V is suitable for 2-mm wire electrodes. The carbon dioxide pressure after the reducing valve should be 0.5 atm; the gas consumption ranges from 1000 to 1500 litres per hour.

The shortest possible arc must be maintained in this process to obtain dense, poreless metal in the weld and to reduce spatter in welding. Welding is done with a constant rate of electrode feed into the arc, the latter being self-regulated. To obtain a stable arc under these conditions the source of power supply (welding converter) should have a flat external characteristic.

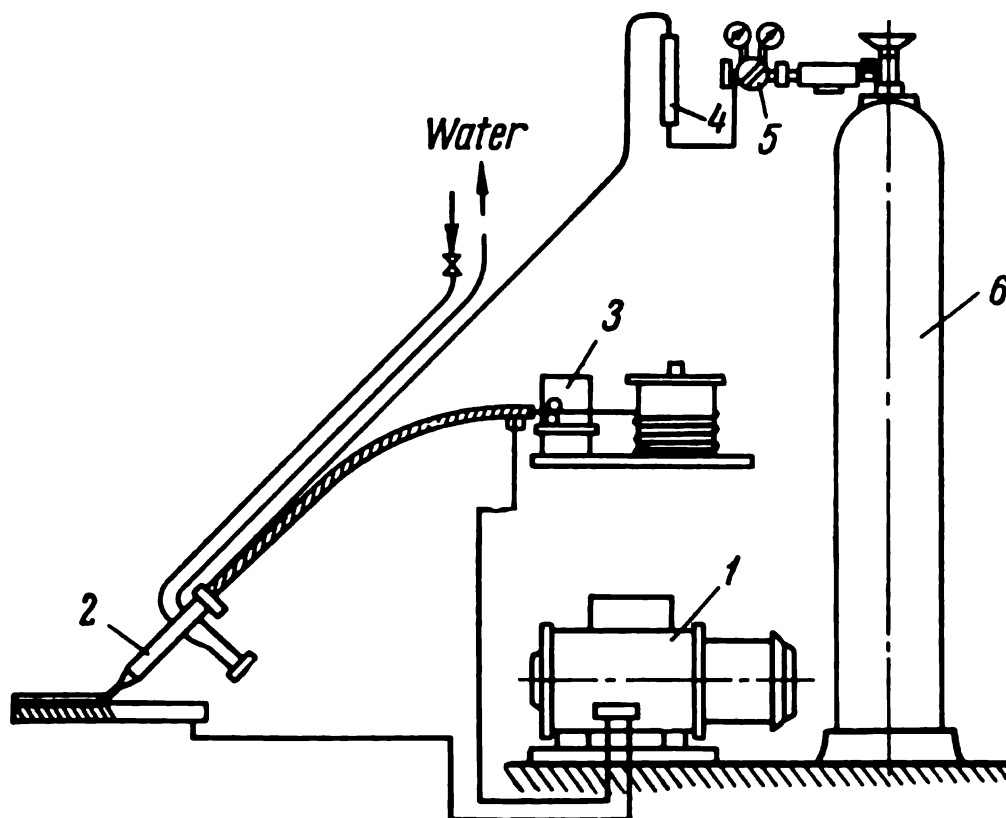


Fig. 167. Equipment for semiautomatic carbon-dioxide-shielded arc welding:

1—welding converter; 2—electrode holder (torch); 3—wire feed mechanism;  
4—gas flow meter; 5—reducing valve; 6—cylinder of carbon dioxide

The arc is established at a voltage of 20 V and higher. A short arc is more stable with reversed polarity.

A steep drooping characteristic can be obtained on the model ПС-500 welding converters by disconnecting the field winding from the brushes and bringing the ends out on a terminal board. This winding is supplied from a separate selenium rectifier. A 24-V generator of the type used for electroplating baths can also be employed in this process.

## Chapter 24

### RESISTANCE WELDING

#### 24-1. Types of Resistance Welding

Resistance welding embraces a group of welding processes in which coalescence is produced by heating the faying surfaces of the elements to be joined to a plastic condition and then applying a high pressure (upsetting) which facilitates the penetration of the atoms of each piece of metal into the other piece. The heat is obtained by the resistance of the work to the flow of electric current in a circuit in which the work is a part.

The amount of heat evolved at the contacting area of the elements is determined from Joule's law:

$$Q = 0.24 I^2 R t \text{ cal.} \quad (86)$$

where  $I$  = current, in amperes,

$R$  = resistance of the circuit at the contacting area of the elements, in ohms,

$t$  = time during which the current flows, in seconds.

A special feature of resistance welding is the rapid heating of the surfaces being welded (in hundredths of a second) due to the application of currents of high amperage.

Resistance welding is a high-production process; it can be easily mechanised and automated, and therefore finds extensive application in construction and the engineering industries.

The principal types of resistance welding are butt, spot and seam welding.

#### 24-2. Resistance Butt Welding

Resistance butt welding may be subdivided into upset, continuous flash and intermittent flash welding processes.

In *upset butt welding* (Fig. 168a) the ends of the parts are brought into contact and heated by the current to a plastic state; then the current is switched off and the metal is upset by the pressure applied between the abutting surfaces. This technique is employed for joining parts of low-carbon steel or nonferrous metals having clean faying surfaces not over 1000 sq mm in area.

In *continuous flash butt welding* the parts to be welded are brought into contact with the current switched on. After heating, the metal is upset with a pressure of from 250 to 500 kg per sq cm. This squeezes out liquid metal and oxides formed on the joint surfaces and the

abutted parts are welded together. This technique is used for joining thin-walled tubes, rails, chain links, tools, press-manufactured parts, as well as combinations of different metals, such as steel-copper-brass, aluminium-copper, etc. Advantages of the process are high productivity and high quality of the welded joint while one disadvantage is the loss of metal in burning and in the flash.

*Intermittent flash butt welding* is performed by alternating tight and lighter contact of the parts to be joined by varying the pressure applied during the welding process. When the required stage of fusion

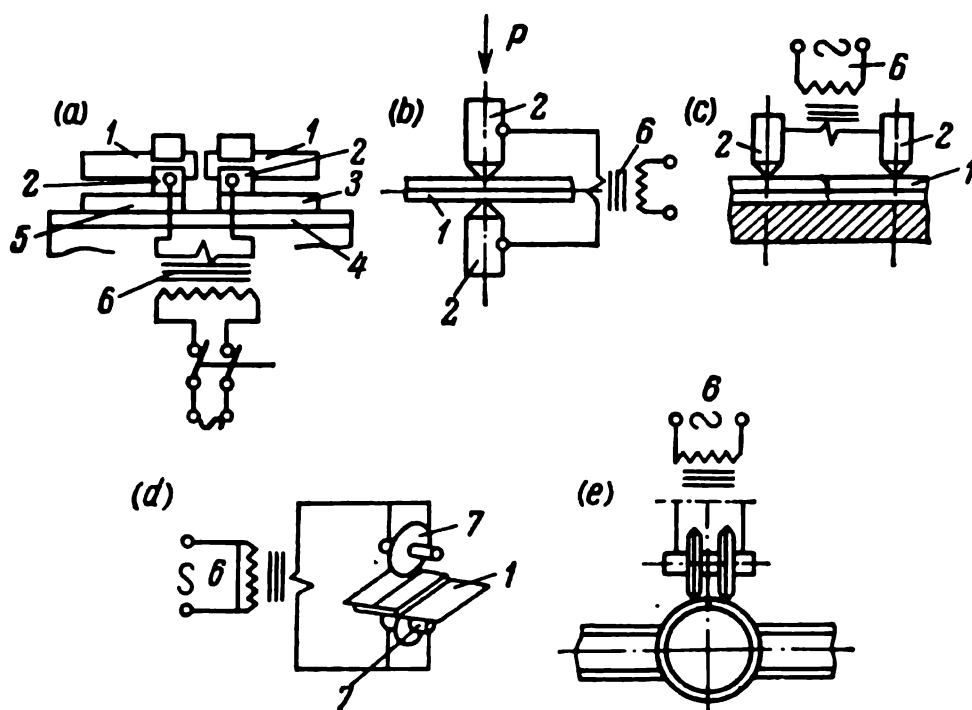


Fig. 168. Basic types of resistance welding:

(a) butt welding; (b) direct spot welding; (c) indirect spot welding; (d) and (e) seam welding; 1—parts being welded; 2—copper clamps; 3—movable slide; 4—guide plate; 5—stationary plate; 6—transformer; 7—rotating roller

is attained the surfaces being welded are forced together sharply with a high pressure. This method will prove expedient in cases when the capacity of the equipment employed is insufficient for continuous flash welding.

The capacity of resistance butt welding machines is selected on the basis of from 0.12 to 0.15 kVA per sq mm of welded cross section. The welding voltage may vary from 5 to 15 V while the pressure required for upsetting ranges from 1.5 to 5 kg per sq mm.

A sound joint can be obtained in butt welding if the abutted surfaces of the parts being welded are equal in cross-sectional area and parallel to each other.

In accordance with their capacity and purpose butt welding machines may be either hand-operated or automatic. The latter type clamp

the parts to be welded and apply pressure for upsetting by electric, hydraulic or pneumatic means, and the process is controlled automatically.

Resistance butt welding is applicable for cross sections of various area and shape—round, square, or irregular. In joining pressworked sheets the length of the weld may reach 2 m. The weld strength in butt welding is not inferior to that of the parent metal and therefore the process is suitable for critical joints.

### **24-3. Resistance Spot Welding**

In spot welding the parts are lapped and clamped between two bar-type metal electrodes connected to the secondary winding of a step-down transformer.

When the current is switched on the lapped pieces of metal are heated in a restricted area. The generated heat melts the surface layer of metal in the central, more highly heated, area of contact with the electrodes, and the adjacent layers of metal are softened to a plastic state. Then the current is switched off and the electrodes are compressed. The spot weld produced has a cast structure and is lentil shaped.

The electrodes used must possess high electrical and thermal conductivity and retain the required strength at temperatures up to 400°C. Cold-rolled electrolytic copper, tungsten-base alloys and cadmium bronze are suitable materials for making electrodes. Electrodes are usually of hollow design and are cooled with water during operation.

Spot welding may be either direct (Fig. 168*b*) or indirect (Fig. 168*c*).

Spot welding machines are available for either mechanised (pedal-actuated) or automatic operation. In the latter case, the current is switched on and off by means of special mechanical, electrical or ionic interrupters. The machines may be of either stationary or portable design. The stationary type are available as single-spot or multi-electrode machines, the latter being capable of welding up to 50 spots simultaneously and having a production capacity up to 200 spot welds per minute.

Spot welders have ratings up to 600 kVA, a current density of at least 120 A per sq mm, secondary voltage from 1 to 12 V, and the pressure developed between the electrodes ranges from 2 to 12 kg per sq mm.

Spot welding is the most widely employed of the resistance welding processes; it finds extensive application in the aircraft and automobile industries for manufacturing airframes and bodies, as well as in railway car building and the instrument industry, for making reinforcement in concrete construction, etc.

#### 24-4. Resistance Seam Welding

Seam welding is used to join sheet material with a total thickness up to 4 mm. The lapped parts prepared for welding are passed between rotating rollers—the electrodes of the seam welder—through which a current flows to produce the weld (Fig. 168d).

Seam welding may be either intermittent or continuous; in the former case a current interrupter is employed. The rollers are made of the same materials as the electrodes for spot welding. Rollers may vary in diameter from 40 to 350 mm depending upon the curvature of the work being welded. Welding currents range from 2000 to 5000 A while the force applied to the rollers may be as high as 500 or 600 kg. Welding speeds from 0.5 to 3.5 m per min are regular practice.

Seam welding is used in the manufacture of tanks, tubes and other articles of steel and nonferrous metals which must have airtight joints (Fig. 168e).

### Chapter 25

#### GAS AND THERMIT WELDING

##### 25-1. Gases Used in Welding

The heat generated by the combustion of various gases in a stream of oxygen is used in gas welding to heat a restricted area of the components to be welded. Combustion of this type produces a flame with a very high temperature. At the moment of fusion of the base metal a rod of filler metal is introduced into the flame; the rod also melts and, together with the molten base metal, forms the welded joint.

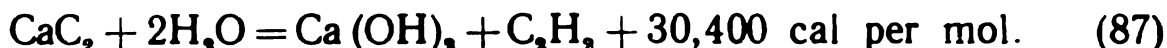
Gas welding is widely used in repair work and in manufacturing thin-walled parts of steel and nonferrous alloys. The fuel gases used in these processes are acetylene ( $C_2H_2$ ), hydrogen ( $H_2$ ), natural gas containing about 94 per cent methane ( $CH_4$ ), petroleum gas, and vaporised gasoline and kerosene.

The most commonly used in welding practice is acetylene ( $C_2H_2$ ) which produces the highest flame temperature ( $3150^\circ C$ ) and generates the greatest amount of heat (11,470 cal per cu m). These data refer to combustion in pure oxygen. Other gases may be employed for cutting metals where a less powerful source of heat can be utilised.

*Acetylene* is lighter than air or oxygen. The presence of acetylene in air in amounts from 2.8 to 65 per cent forms an explosive mixture. Acetylene is ignited at a temperature of  $420^\circ C$  and becomes explo-

sive at pressures over 1.75 atm or after prolonged contact with copper or silver.

Acetylene is generated by the interaction of calcium carbide and water. A great deal of heat is evolved in this reaction:



Calcium carbide is supplied, as a rule, in steel drums weighing up to 100 kg and is in the form of granulated lumps from 2 to 80 mm in size. The average yield of acetylene per kilogram of commercial carbide varies from 250 to 300 litres.

Acetylene for welding is supplied either from generators, where it is obtained from calcium carbide, or it may be purchased in metal cylinders in which it is dissolved in acetone under a pressure of from 16 to 22 atm gauge (due to the danger of explosions pure acetylene cannot be compressed to more than 1.5 atm). At normal pressure one litre of acetone dissolves about 24 litres of acetylene. As a safety measure the cylinder of acetylene is filled with a porous filler (usually charcoal) forming a system of capillary vessels.

The oxygen required for the welding flame is made by liquefying air (cooling it to  $182.8^\circ\text{C}$  below zero) and separating the oxygen from the nitrogen. One litre of liquid oxygen, a transparent bluish substance, yields 790 litres of gaseous oxygen at a pressure of 760 mm mercury column.

Liquid oxygen is transported in special tanks (vacuum flasks) by automobile or railway and is poured into gasifiers at the site of operation.

Gasifiers may be either of the heated or cold types. In the former, the liquid oxygen is heated and vaporised while in the cold gasifiers it is stored in the liquid state under a pressure of 15 atm. The liquid oxygen is vapourised as needed in piping, compressed to 150 atm and stored in steel cylinders.

## 25-2. Structure of the Oxy-acetylene Welding Flame

The complete combustion of acetylene in an atmosphere of oxygen can be represented by the following summary reaction:



Actually, the burning of the acetylene in the flame of the welding torch comprises three elementary processes:

- (1) decomposition of acetylene into elements  $\text{C}_2\text{H}_2 \rightleftharpoons 2\text{C} + \text{H}_2$ ;
- (2) first stage of combustion of the acetylene decomposition products—incomplete combustion of the carbon  $2\text{C} + \text{O}_2 \rightleftharpoons 2\text{CO}$ ;
- (3) second stage of combustion of the acetylene decomposition products consisting of the oxidation of CO to  $\text{CO}_2$  and  $\text{H}_2$  to  $\text{H}_2\text{O}_{\text{vapour}}$ .

This can be expressed by the reaction  $2\text{CO} + \text{H}_2 + 1.5\text{O}_2 \rightleftharpoons 2\text{CO}_2 + \text{H}_2\text{O}_{\text{vapour}}$ .

The welding flame is shown schematically in Fig. 169. It consists of three zones: inner luminous cone 1, reducing zone 2 and oxidising zone 3.

The first of these zones has the shape of a truncated cone with a rounded end. This zone consists of the partially decomposed products of acetylene with the separated solid particles of carbon. The latter are incandescent and glow with a blinding white luminescence. This is the brightest part of the flame.

Heat is evolved in the reducing zone 2 mainly due to the oxidation of the incandescent particles of carbon in carbon monoxide. The highest temperature in this zone (up to  $3000^\circ\text{C}$ ) is produced at a distance of 3 to 5 mm from the end of the inner cone. The products of acetylene combustion, CO and  $\text{H}_2$ , in the reducing zone can reduce oxides, including iron oxides formed in welding.

Afterburning of CO to  $\text{CO}_2$  and  $\text{H}_2$  to  $\text{H}_2\text{O}_{\text{vapour}}$  takes place in the oxidising zone 3 with a surplus of oxygen from the air.

The gaseous products that enter this zone together with the oxygen (mixture of  $\text{CO}_2$ ,  $\text{H}_2\text{O}_{\text{vapour}}$  and  $\text{O}_2$ ) have an oxidising capacity though they form a shield preventing direct contact between the molten metal and the air.

The following types of oxy-acetylene flames are obtained depending upon the oxygen-acetylene ratio in the initial gaseous mixture:

(1) Neutral flame, when the ratio is  $\text{O}_2 : \text{C}_2\text{H}_2 = 1$  to 1.2 (see Fig. 169).

(2) Carbonising, or reducing, flame at ratios of  $\text{O}_2 : \text{C}_2\text{H}_2 < 1$  when there is an excess of acetylene. The inner cone is longer in this case than in a neutral flame and the flame itself loses its sharp outline. Such flames are used in welding cast iron and in hard surfacing with high-speed steel and cemented carbides.

(3) Oxidising flame at ratios of  $\text{O}_2 : \text{C}_2\text{H}_2 > 1.3$ , i. e., when oxygen is in excess. In this case, the flame acquires a light-blue tint and the inner cone is shorter than in the neutral flame.

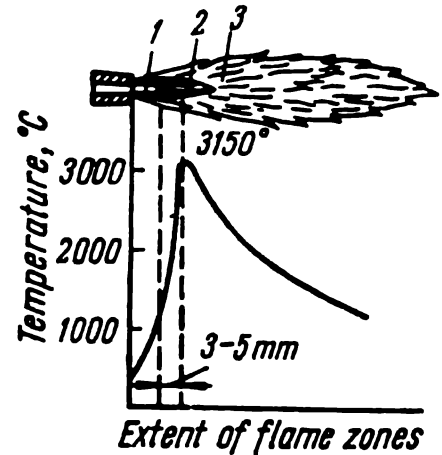


Fig. 169. Structure of the oxy-acetylene welding flame

### 25-3. Gas Welding Equipment

Equipment required in gas welding includes acetylene generators, cylinders for compressed gases, regulators and welding torches (blowpipes).



Acetylene is usually produced in *generators* in which calcium carbide is decomposed by water.

According to the method by which the calcium carbide reacts with the water, acetylene generators are classified into three systems, characterised by the design of the gas-forming device:

(1) Carbide-to-water system (Fig. 170a), in which portions of calcium carbide are dropped periodically into a large volume of water. These generators provide the highest acetylene yield from the carbide—about 95 per cent.

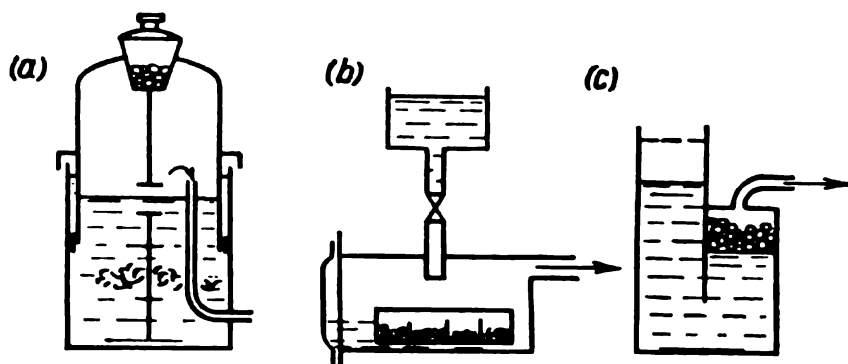


Fig. 170. Types of acetylene generators

(2) Water-to-carbide system (Fig. 170b), in which the water is periodically fed to the carbide in a box open at the top. This box is enclosed in an airtight retort. Such generators have a lower acetylene yield—85 to 90 per cent.

(3) Contact system (Fig. 170c), in which calcium carbide and water are periodically brought into contact and separated again in accordance with the acetylene consumption from the generator. The contact system is the poorest insofar as acetylene yield is concerned and is employed for portable low-output generators exclusively.

Industrial generators are of the low-pressure (up to 0.1 atm gauge) and medium-pressure (up to 1.5 atm gauge) types. The medium-pressure variety are more convenient since the composition of the flame can be more easily regulated and more constant welding conditions are maintained. High-pressure generators (over 1.5 atm gauge) are not employed in industry.

The type ΓA low-pressure acetylene generator shown schematically in Fig. 171 is the most widely used in Soviet industry. It has an output of 1.2 cu m per hr and yields gas under a pressure up to 0.014 atm gauge. The housing 1 of the generator is filled with water through funnel 2. The weight of bell 3 floating in the water provides the required gas pressure. Calcium carbide is loaded into box 4 which is placed into retort 5 having an airtight cover 6. The amount of water admitted to the retort through pipe 7 is regulated by cock 8. Acetylene formed in the retort passes through pipe 9 under hood 10.

From here it passes farther through pipe *11* to the gas cleaner *12* and then through the water seal *13* to the hose connected to the welding torch. Safety pipe *14*, welded to the bell, serves to release excess acetylene to the atmosphere. Acetylene is usually supplied to the welding stations directly from the generator; only in special cases it is delivered to the welding site in the compressed state in cylinders.

Safety *water seals* may be of either low-, medium- or high-pressure design. The principle of the low-pressure seal (Fig. 172) is as follows.

Acetylene enters seal *1* through gas-input pipe *2*. The seal is filled with water up to the level of test cock *4*. After passing through a

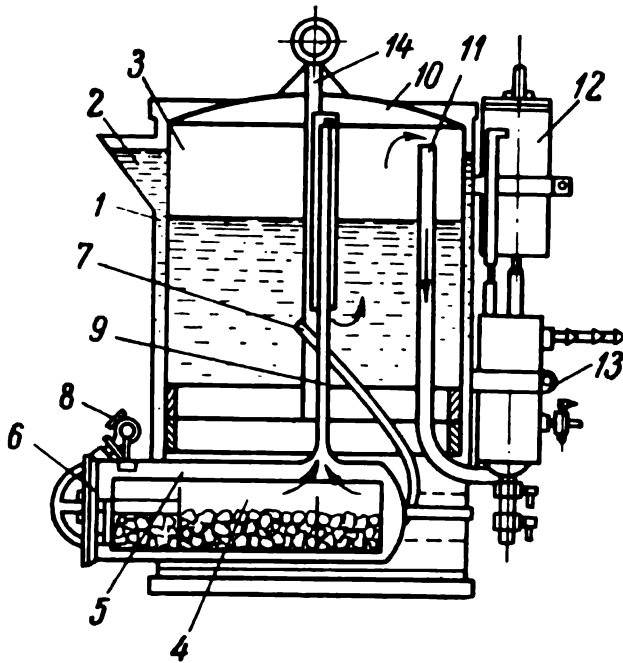


Fig. 171. Low-pressure acetylene generator

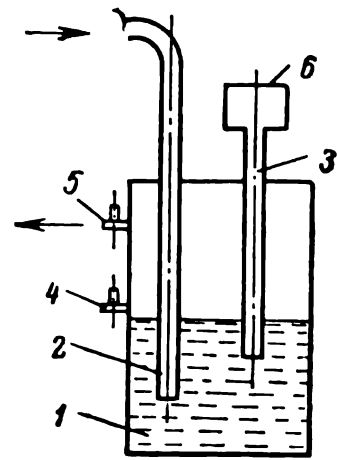


Fig. 172. Principle and operation of a water seal

layer of water, the acetylene is delivered to the torches through cock *5*. The working pressure in the gas torch is determined by the difference in water level in safety pipe *3*, which is open on top, and in the seal. In case of a backfire the gas mixture enters the seal through cock *5* forcing the water up into gas-input pipe *2*. This lowers the water level in the seal so that the lower end of the safety pipe is exposed and the gases are released to the atmosphere. The water plug, thus formed in gas-input pipe *2*, prevents the blast from entering the generator. Shield *6* returns the water to the seal, thereby reducing its loss.

*Cylinders for compressed gases* (Fig. 173) are seamless steel vessels *1* of cylindrical form at the bottom of which a steel shoe *2* is fitted. The upper part of the cylinder ends in a semispherical neck *3* on which a safety cap *4* is screwed. The neck has a hole with taper thread *5* into which valve *6* is screwed to regulate the discharge from the cylinder.

Oxygen cylinders are painted blue; those for acetylene are painted white. Standard cylinders have a capacity of 40 litres.

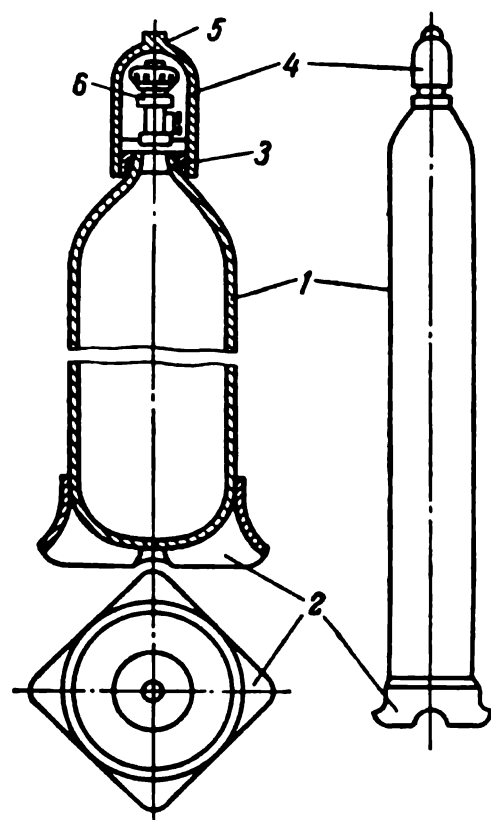
*Regulators* serve to reduce the cylinder pressure of the gas to a suitable working value and to maintain a constant pressure. Depending upon the number of stages of reduction regulators may be of single- and two-stage construction. The two-stage type has a lower pressure drop in each stage and can reduce the pressure to a lower value. In the single-stage design the gas pressure is reduced in one

chamber; the two-stage regulator reduces the pressure in two chambers consecutively. In the first chamber the pressure is reduced to 40 or 50 atm gauge and in the second, to the working value.

A two-stage regulator is illustrated in Fig. 174. The device is connected to the cylinder by means of a captive nut 1 which is screwed onto the connection of the valve. When the valve is opened the gas passes through the hole in connection 2 after passing through pipe 3 which contains several layers of fine brass wire mesh.

The connection contains the heat absorber in the form of red copper plates 4 with several holes. The gas passes from the absorber into body 5 of the regulator where its pressure is indicated by pressure gauge 6. In this part of the device the pressure is equal to that in the cylinder or gas main. The gas next passes to the reducing valve 7. On entering the first reducing

Fig. 173. Cylinders for compressed gases



chamber 8 the gas expands and its pressure drops to 25 or 35 atm gauge. The pressure in this chamber is set by means of regulating nut 9 which compresses the main spring 10. The latter pushes the pressure disk 11, diaphragm 12 and transmitting disk 13 upwards. On the top of disk 13 is steel pin 14 which lifts reducing valve 7. Valve 7 is held against its seat by the action of closing spring 15.

After passing through reducing valve 16 of the second stage, the gas enters the second reducing chamber 17 where it expands until its pressure reaches the working value. This pressure can be regulated by means of screw 18 in the range from 1 to 15 atm gauge. The working pressure is indicated by pressure gauge 19 having a scale of 40 atm gauge, graduated in 2.5 atm divisions. Movement is transmitted from regulating screw 18 to reducing valve 16 in the second chamber through main spring 20 in the same manner as in the first chamber.

The regulator also has a safety valve 21 which opens at a pressure of 45 to 50 atm gauge, releasing excess gas to the atmosphere. Valve 22, mounted on the gas-output connection, is resorted to when the gas is to be shut off for a short time.

Gas is delivered from the regulator to the welding torch.

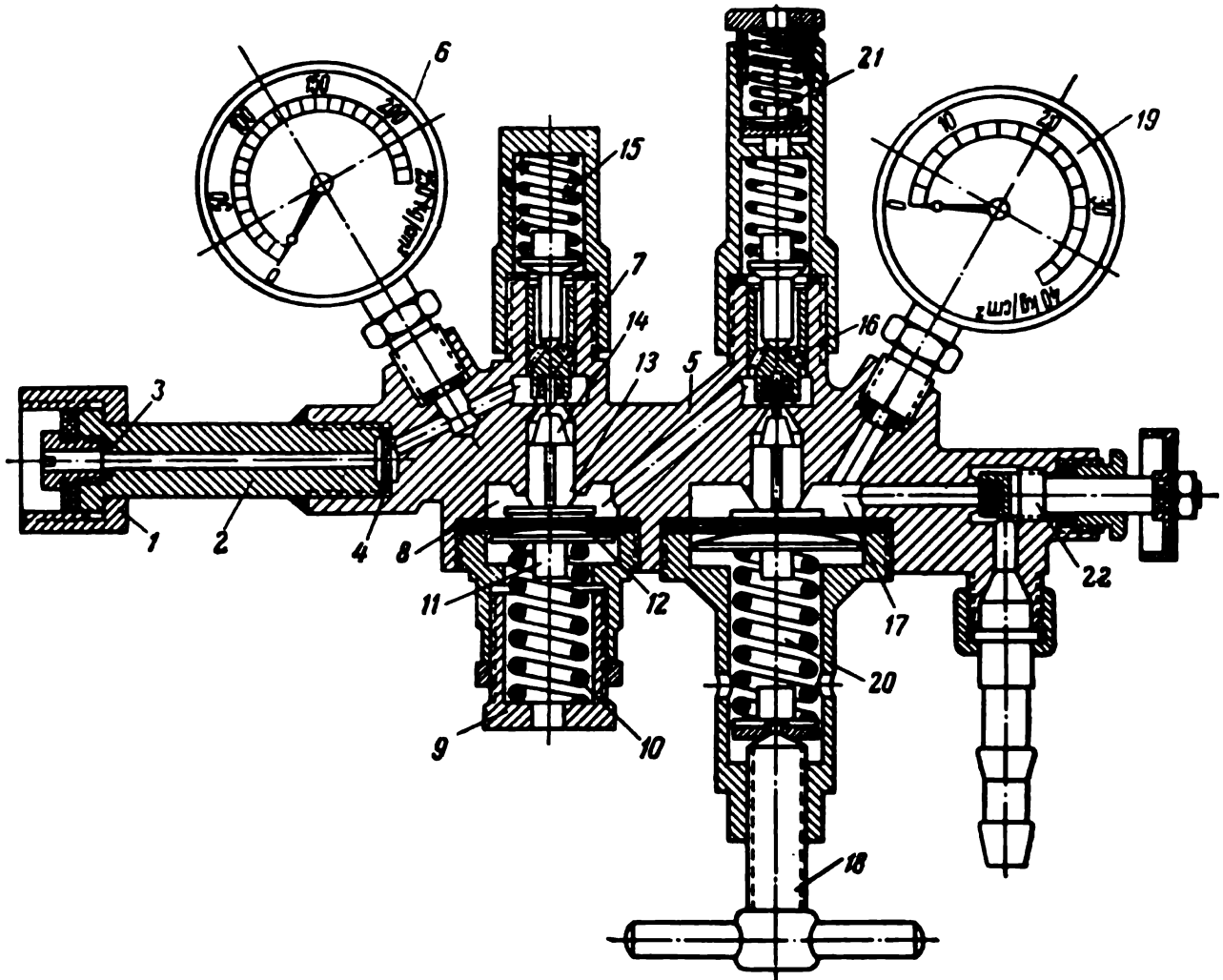


Fig. 174. Principle of a two-stage regulator

The *welding torch*, or blowpipe, serves to mix the gases and to deliver the mixture to the tip where it is burned. Depending upon their principle of operation, torches are classified as the injector type, operating with a low gas pressure, and the medium- and high-pressure types. As to their purpose, torches may be of the welding, cutting, brazing, surface-hardening and other types. Both single- and multiple-flame torches are used.

Injector torches are the more widely used in construction and industry.

The type  $\Gamma C$  injector torch is shown in Fig. 175. Oxygen enters the torch through a rubber hose, fitted on hose stem 1, and pipe 3;

acetylene, through stem 2 and hollow handle 4. Valves 5 and 6 control the volumes of gases admitted into the torch.

The torch is furnished with a set of interchangeable heads (from No. 0 to No. 7) for welding metals of various thicknesses. The head is inserted into member 7 on which the captive nut 8 is screwed.

After passing through valve 5 the oxygen enters injector 9 and then mixer 10 where it produces a vacuum in the acetylene channel. From the mixer 10 the gas mixture passes along pipe 11 of the head to the tip 12 and from the tip to the atmosphere.

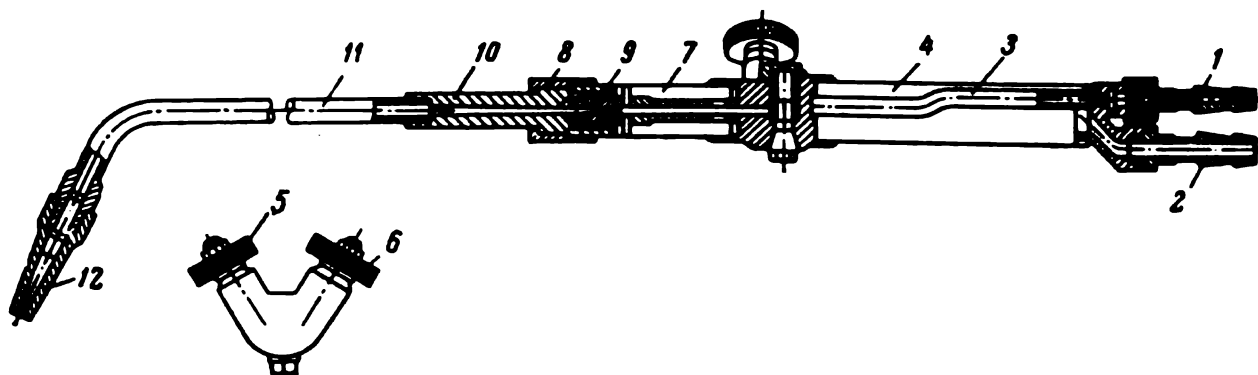


Fig. 175. Injector gas welding torch, type ΓC

Acetylene consumption varies from 50 to 2800 litres per hour; oxygen consumption—from 50 to 3100 litres per hour.

The type ΓC torch can weld steel in thicknesses from 0.8 to 30 mm.

The application of multiple-flame torches with several heads speeds up welding and enables higher-quality welds to be made.

To light a torch the oxygen valve is opened slightly and then the acetylene valve is opened one full turn. The gas mixture flowing from the tip is ignited and the flame is regulated by means of the valves. The flame is extinguished by first closing the acetylene valve and then the oxygen valve.

## 25-4. Gas Welding Techniques

A sound welded joint may be obtained upon proper selection of the torch size, filler material, method of moving the torch along the weld and the angle at which the torch is held, as well as proper regulation of the welding flame.

The size of torch required depends upon the thickness and heat conductivity of the metal being welded. Metals with higher conductivity require a torch head with a larger gas consumption.

The joint is first heated with the torch until the welding pool is formed. Then molten metal from the welding wire or rod is added to the pool.

When relatively thin metal is welded, the so-called forehand welding is used in which the torch flame progresses from right to left. The welding rod is to the left of the torch flame and precedes it.

Metal over 5 mm thick is welded by the backhand technique in which the torch progresses in front of the rod from left to right. Backhand welding is faster by 20 to 25 per cent and from 15 to 25 per cent less acetylene is needed.

The angle at which the torch is inclined to the surface being welded depends upon the thickness of the metal. Thicker metal requires a higher concentration of heat and, consequently, a larger torch angle.

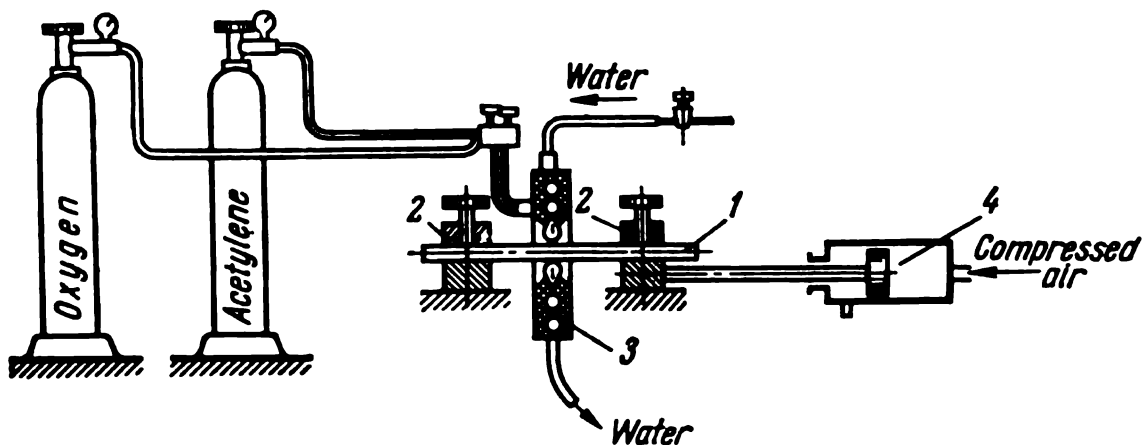


Fig. 176. Principle of the pressure-gas welding process

Horizontal and overhead welding is usually done by the backhand technique while vertical joints are welded upwards by the forehand method.

The welding wire or rod used as filler material in gas welding should have a chemical composition similar to that of the parent metal.

The diameter of the filler rod is selected to suit the thickness of the parent metal. The following empirical formula can be employed for approximate determination of the rod diameter:

$$d = \frac{s}{2} + 1, \quad (89)$$

where  $d$  = rod diameter, in mm,

$s$  = thickness of the parent metal, in mm.

The application of a flux improves the quality of gas welding since it protects the weld against oxidation.

**Pressure gas welding.** This is a gas welding process in which the abutting surfaces of the parts being welded are heated by an oxy-acetylene flame to a state of fusion or plasticity and then coalescence is produced by the application of a mechanical force.

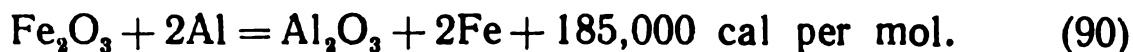
The principle of pressure gas welding is illustrated in Fig. 176. The parts 1 to be welded are clamped in the special fixture 2; the abutting ends are rapidly heated over a restricted area by means of a multiple-flame circular torch 3 whose shape depends upon the section of the elements being welded. The torch is water cooled. Upsetting is done by the action of the pressure mechanism 4 after the joint reaches the welding heat and the metal is brought to the plastic condition. An oscillating motion of 10 mm to each side is imparted to the torch to ensure more uniform heating of the faying surfaces on the parts.

Pressure gas welding finds wide application for joining pipeline mains, rails, tools, etc.

### 25-5. Thermit Welding

Thermit welding makes use of the heat supplied by the exothermic reaction involving the combustion of thermit which is a mixture of powdered aluminium (20 to 22 per cent) and iron oxide in the form of roll scale (80 to 78 per cent).

Thermit burns according to the following reaction:



The reaction will proceed if the thermit is preheated at least in one place to 1200°-1300°C by means of a primer or ignition powder. A temperature of 3000°C may be attained in the combustion of the thermit mixture. The reaction results in the formation of liquid reduced iron and slag.

Thermit welding processes may be classified as: (1) the non-pressure method, in which no mechanical force is applied; (2) pressure method with the application of a force and (3) combined method, in which coalescence is achieved by applying pressure after the parts are heated to a state of fusion.

In all of the thermit welding processes a two-piece refractory mould is fitted over the joint in which the abutting surfaces have been preliminarily cleaned. The chemical reaction occurs within a special crucible and the molten material is allowed to pour into the mould. This material either melts the abutting surfaces or brings them to a plastic state.

Thermit welding is used in joining rails and pipes, and in repairing heavy parts.

## Chapter 26

### WELDABILITY OF METALS

#### 26-1. Weldability of Carbon Steels

Decisive factors affecting the weldability of steels are their chemical composition, physical properties and the heat treatment to which they are subjected.

*Carbon* has a strong influence on the quality of the weld. An increase in the carbon content affects the strength, hardness and toughness of the weld. When the carbon content exceeds 0.3 per cent the air-hardening capacity of the transition zone in the base metal is increased and the steel becomes more brittle. The effect produced by carbon is much less in arc-welding than in gas-welding process.

Carbon structural steels have good weldability with any welding technique for carbon contents up to 0.27 per cent; satisfactory welds can be obtained for contents up to 0.35 per cent. Welds of the highest quality can be made in these steels with proper selection of the welding conditions and if reliable protection is ensured against the action of the air on the molten metal.

Structural steels, containing more than 0.36% C and susceptible to the formation of hardening cracks, require preheating before welding and subsequent heat treatment. The application of filler metal with a low carbon content enables hardening of the weld to be avoided; weld strength can be ensured by alloying the metal of the weld with manganese, silicon and other elements.

*Phosphorus* in steel, in amounts exceeding 0.04 per cent, has an unfavourable effect on the mechanical properties of the weld since the brittleness is increased.

*Sulphur* has an extremely detrimental effect on the weldability of steels since it causes red shortness. The formation of cracks is observed at a sulphur content over 0.04 per cent (especially in gas welding).

#### 26-2. Welding of Alloy Steels

Various difficulties arise in the welding of alloy steels. These include the burning out of the alloying elements, precipitation of carbides in heating, air-hardening of the deposited metal and the transition zone, and the development of shrinkage stresses and the formation of cracks due to the low heat conductivity of certain grades of alloy steel.



The above-mentioned troubles can be prevented or eliminated in welding alloy steels by not allowing their overheating, by strictly observing the prescribed welding conditions, by using fluxes and coatings of special composition, by preheating the work before welding and by heat treatment of the work after welding.

The weldability of low-alloy pearlitic structural steels with a total alloying element content of 3 or 4 per cent is determined mainly by the carbon content. The weldability of such steels is good at a carbon content up to 0.2 per cent, fair at 0.35% C, limited at 0.4% C and very poor at a content over 0.4 per cent. Sound welds can be made only by preheating and subsequent annealing.

The weldability of chromium stainless steels containing from 12 to 14% Cr is quite satisfactory if the carbon content does not exceed 0.2 per cent. Precautions must be taken in welding these steels to prevent the burning out of chromium by using protective electrode coatings containing chromium. It is advisable to weld chromium steels by preheating the metal before welding and subjecting it to subsequent heat treatment. Chromium steels become brittle after welding and, consequently, the welds are not capable of withstanding heavy deformation. The ductility of the welds can be restored by annealing at a temperature of 760° C.

Chromium steels containing up to 17% Cr and more belong to the ferritic class. It is advisable to weld such steels only after preheating to about 200° C since this increases the ductility of the metal.

Chromium steels containing from 25 to 30% Cr are quite weldable but the metal of the weld acquires low mechanical properties because of the intense grain growth at the welding temperature and the development of internal stresses. Annealing at a temperature of 950°-980° C is applied after welding to relieve the internal stresses.

Grain growth can be avoided in welding by the addition of titanium, vanadium and aluminium to the material of the electrodes or their coating.

The welding of chromium steels of all grades is done with electrodes whose composition is the same as that of the base metal and which are coated with a special compound. This coating prevents the chromium from being burned out and saturates the welding pool with this element.

No difficulties are encountered in welding austenitic chrome-nickel and stainless steels containing 18% Cr and 8% Ni.

In welding austenitic manganese steel containing from 0.8 to 1.3% C and 12 to 14% Mn, the carbides precipitate from the austenite and even a partial transformation of austenite to martensite is possible in the heat-affected zone of the weld. This sharply reduces the ductility of the metal and may be the cause of crack formation.

To avoid cracks welding must be done as rapidly as possible and certain sections of the weld must be cooled by water.

Manganese steels are welded with electrodes of the same composition as the base metal or of manganese-nickel steel with a special coating.

### 26-3. Welding of Cast Irons

The structural features of cast iron (the presence of free graphite) and the high carbon content lead to considerable difficulties in welding. Furthermore, cracks may form in cast iron upon heating since its ductility is low.

Cast iron may be welded either hot or cold.

*Hot welding* is performed by preheating the work to 600°-650°C. This temperature is maintained throughout the whole welding process. A considerable volume of liquid metal is produced at the weld. This retards the cooling of the molten metal and thereby prevents chilling. Retarded cooling ensures more complete graphitisation of the cast iron, prevents crack formation and enables a weld of higher density to be made. In properly made welds the deposited metal acquires the structure of grey cast iron with well-formed graphite inclusions.

The hot welding of cast iron may be carried out with either an electric arc or a gas flame and with either complete or restricted preheating of the work. Complete preheating is applied for welding small parts of intricate shape and is accomplished in metal boxes heated by charcoal.

After making the weld the part is covered with asbestos or coal dust to retard cooling.

Cast-iron welding rods, containing up to 3.6% Si, are used as filler metal. The flux is either calcined borax or a mixture of borax (50 per cent), sodium bicarbonate (47 per cent) and silica (3 per cent).

The welding flame should be either carbonising or with a slight excess of acetylene. The intensity of the flame is selected to suit the thickness of the metal on the basis of a 100- to 120-litre per hour acetylene consumption per millimetre of work thickness.

The more widespread method is *the cold welding* of cast iron with an electric arc. No preheating is resorted to in this process. The electrodes employed are cast-iron or steel rods with a special coating, rods of copper and nickel alloys (Monel metal) and others. Monel metal (70% Ni and 30% Cu) is used to repair various defects in grey-iron castings by welding.

Since the temperature in welding cast iron does not exceed 30°-40° C the cold welding process should be done with a short weld. The finished weld is covered with sheet asbestos to retard cooling.

Excessive hardness of the metal at the weld can be avoided by the use of composite electrodes, for example lightly coated steel electrodes, from 3 to 5 mm in diameter, inserted into a copper tube.

To obtain a weld with high mechanical properties, it is advisable to set threaded steel studs at the weld to transmit the thermal stresses to the whole of the work.

The welding of malleable iron is much simpler since it possesses a certain degree of ductility which excludes crack formation. If the part of malleable iron is to undergo annealing after the welding operation, a steel electrode or welding rod is employed for arc or gas welding; if not, then gas welding may be used with a brass filler rod or arc welding with Monel metal electrodes.

Much more complicated is the welding of high-strength cast iron. In the first place, it is difficult to obtain a weld with high mechanical properties, and in the second, these cast irons have an increased susceptibility to hardening and chilling.

Defects in grey-iron castings may be repaired by means of gas welding, using rods of magnesium cast iron. Small defects can be welded over in such castings by using arc welding with iron-nickel electrodes.

#### **26-4. Welding of Aluminium and Aluminium Alloys**

There are many unfavourable factors affecting the welding of aluminium and its alloys. Aluminium oxidises easily and the film of aluminium oxide that forms on its surface has a melting point of 2050°C (higher than the boiling point of aluminium, i. e., 1800°C) and a specific weight of 3.85 g per cu cm (the specific weight of aluminium is 2.7 g per cu cm). This film protects the surface of the aluminium against further oxidation but also impedes fusion of the edges. In heating to the melting point, aluminium passes rapidly from the solid to the liquid state. This is often accompanied by the buckling or sagging of whole areas of the work or the formation of fractures.

Aluminium and its alloys possess high heat conductivity and a high latent heat of fusion; therefore, they require more heat for welding.

Hydrogen is quite soluble in liquid aluminium but its solubility decreases with a fall in temperature and it separates from the metal along the grain boundaries in the form of fine bubbles which weaken the weld and the near-weld zone.

The shrinkage of aluminium and its alloys upon solidification is quite large and therefore cracks may form near the weld. Welding rods and electrodes of a special composition are employed to avoid large shrinkage and cracking in welding.

Taking into account these features of aluminium, the edges of aluminium parts and the filler rods are cleaned of dirt with a wire brush before welding, degreased with gasoline or a caustic soda solution and subjected to pickling. Pickling is done in a solution of commercial caustic soda—45 to 55 g, commercial sodium fluoride—40 to 50 g and water—1 litre. The temperature of the pickling bath ranges from 50° to 70°C while the pickling time is one minute. After pickling the parts are washed consecutively in cold and hot water.

Work of aluminium and its alloys is welded by the techniques described above.

Sheets up to 4 mm thick are butt welded without bevelling the edges and with a clearance of 0.5 mm. A single-V butt joint is suitable for sheet from 4 to 18 mm in thickness and a double-V joint with the edges bevelled from 30° to 35° for thicker material. Thin sheets can be welded without filler material, flanging the edges for this purpose.

Due to the intensive oxidation in welding aluminium and its alloys, it is advisable to use a flame with only small excess of acetylene. The intensity of the flame should be somewhat higher than for welding steel.

Powdered fluxes or pastes of special composition are used to dissolve the aluminium oxide film and to remove it from the welding pool. The most widely used is the Soviet flux AΦ-4a which contains 50 per cent potassium chloride, 28 per cent sodium chloride, 14 per cent lithium chloride and 8 per cent sodium fluoride. After welding, the slag and the remaining flux are removed from the surface by scrubbing with warm water, then a 5 per cent solution of nitric acid with 2 per cent potassium bichromate and again with water for 5 minutes. Finally, the washed weld is dried.

Wire of the same chemical composition as the work is used as the filler material. Good results are achieved in welding the aluminium alloy, grade AMu, and certain heat-treatable alloys by the use of filler wire, grade AK, containing about 5% Si. This material improves the fluidity of the molten metal in the weld and reduces its shrinkage during cooling.

Aluminium castings are welded after preheating to 250°-260° C; welding is sometimes followed by annealing at 300°-350° C to obtain a fine-grained structure in the weld and to relieve internal stresses.

A special coating is applied in arc welding with a metal electrode. It contains up to 15 per cent sodium chloride, up to 50 per cent potassium chloride and up to 35 per cent cryolite. To 100 g of this dry mixture, 50 cu cm of water is added. Here, the binding material is the sodium chloride which, at the same time, provides the fluxing

action. The coat applied to the electrode is from 1 to 1.2 mm thick.

Welding is performed with direct current and reverse polarity.

Carbon-arc welding is done with filler metal and flux of the same compositions as used in gas welding.

Atomic-hydrogen welding of aluminium and its alloys is employed in critical structures for joining parts from 1.5 to 10 mm in thickness. The filler metal and flux are the same as for gas welding.

Argon-arc welding with a consumable electrode is used for parts from 4 to 100 mm thick or thicker; welding is done with a nonconsumable electrode for thicknesses from 0.5 to 15 mm. No fluxes are required in this process, therefore sheets may be welded with butt, tee, lap and other joints. Argon-arc welding of aluminium may be recommended for critical structures subject to heavy loads or requiring airtight joints.

Heavier currents are used in resistance welding of aluminium alloys than for steel of the same thickness but the welding time must be less. This is due to the higher heat and electrical conductivities of aluminium alloys in comparison with steel. For example, in spot welding sheet steel 2 mm thick, a current of 7500 A is recommended with a welding time of 0.5 second and an electrode force of 300 kg while in welding sheet duralumin of the same thickness, the corresponding values are 31,000 A, 0.12 second and 500 kg.

Machines designed for welding aluminium alloys incorporate special ionic interrupters to ensure proper welding time.

### **26-5. Welding of Magnesium Alloys**

Alloys of magnesium with manganese, zinc and aluminium are used in engineering. These alloys are distinguished for their low specific weight (1.76 to 1.8 g per cu cm) and sufficiently high mechanical properties. The melting points of magnesium alloys range from 460° to 650°C. Magnesium actively combines with oxygen and forms a surface film of magnesium oxide, but one that is weaker than the oxide film formed on aluminium. Therefore, such a film of magnesium oxide is poor protection against corrosion of magnesium alloys.

Problems that arise in welding magnesium alloys concern the high inflammability, low melting point and the formation of the very refractory magnesium oxide (melting point—2500° C) on the surface of the welding pool. Magnesium alloys not only oxidise when they are heated, but actively combine with nitrogen, forming magnesium nitride which weakens the weld. Magnesium also dissolves hydrogen which leads to porosity of the weld.

The techniques involved in welding magnesium alloys are similar to those used for aluminium alloys.

## 26-6. Welding of Copper and Copper Alloys

In welding copper and its alloys it is necessary to take into consideration certain features of these metals.

Metallic copper has a very high heat conductivity, 5.5 times higher than that of iron.

When copper is heated above 400° C in an oxidising flame, cuprous oxide ( $\text{Cu}_2\text{O}$ ) is formed. Copper and  $\text{Cu}_2\text{O}$  form a brittle eutectic. Cuprous oxide is readily reduced by hydrogen to form water vapour which is practically insoluble in solid copper. This leads to the development of internal stresses in heating and is the cause of cracks (hydrogen attack).

Copper is welded, in the main, either with a gas flame or a carbon arc.

The filler rod used in gas welding is of pure electrolytic copper or of copper containing up to 0.2% P and up to 0.3% Si. These two elements are intensive deoxidisers of the welding pool.

The most widespread type of flux for welding copper is a mixture of borax (70 per cent), boric acid (10 per cent) and common salt (20 per cent).

Because of the high heat conductivity of copper, the torch used should be from 1.5 to 2 times larger than for welding steel; acetylene consumption ranges from 150 to 200 litres per hour per millimetre of work thickness.

Copper is welded at increased speeds (to avoid overheating the metal near the weld) and with the torch tip inclined at an angle of from 70° to 80° to the work surface. Very thick work must be preheated during welding with a second torch. It is advisable to forge the weld at a temperature of 400°-500° C to improve its mechanical properties, and to anneal the whole workpiece to refine its grain structure.

Both open and submerged-arc techniques are employed in the carbon-arc welding of copper. In the open arc method the filler rods used are of phosphorous copper and have a coating whose composition includes borax, sodium acid phosphate, silicic acid and charcoal. Special fluxes are applied in submerged-arc welding.

A copper electrode can be employed to arc-weld parts of copper, especially sheet copper. In this case, all procedures, including automatic welding, are carried out by the submerged-arc method.

Brass is welded, as a rule, by the oxy-acetylene and carbon-arc processes. Welding is done in the same way as for copper. At the present time an automatic submerged carbon-arc method has been developed for welding brass.

Bronze is welded by the gas, arc and argon-arc methods. At a temperature of 600° C tin bronzes acquire increased brittleness and are, therefore, welded on pads. The purpose of the pads is to dissi-

pate excess heat so as to accelerate cooling of the metal in the welding pool.

Borax is used as the flux in the gas welding of bronze; aluminium bronzes require flux, grade AΦ-4a.

### **26-7. Welding and Brazing of Titanium**

The most commonly used processes for welding titanium are the inert-gas shielded-arc method and arc welding in a chamber filled with an inert gas. Tungsten electrodes and direct current are regular practice for welding titanium and its alloys.

No neutral-gas protective atmosphere is needed in the spot welding of titanium but the surfaces involved must be clean.

In brazing, titanium may be heated either by a gas flame or in furnaces with a controlled atmosphere.

There are many advantages in using the induction heating method in brazing titanium. Induction brazing allows the use of a controlled atmosphere and ensures rapid, uniform, localised heating of the work.

It is difficult to achieve uniform heating of the metal in resistance brazing since the brazing alloy remains unmelted at the edges of the joint while the material at the centre is highly overheated.

Copper and its alloys are not suitable as brazing alloys since titanium forms brittle compounds with copper. The best brazing alloy is silver. Quite satisfactory brazing filler metals for titanium are alloys of the titanium-nickel system, containing from 28 to 66% Ni and having a melting point in the range from 955° to 1110° C.

### **26-8. Hard-facing of Metals**

Hard-facing, or surfacing, involves the application of a weld deposit at a definite place on the work to increase the hardness, strength and wear resistance of the base metal.

Hard-facing materials include stellite, sormite, stalinite and other cutting and wear-resistant alloys.

Hard-facing is done by means of gas, arc or shielded-arc welding techniques.

Gas and shielded-arc welding ensures a more uniform composition of the deposited layer but ordinary arc welding is less expensive.

The thickness of the deposit should not, as a rule, exceed 2 mm; the susceptibility to crack formation increases with thicker layers.

The hard-facing techniques and conditions that are used should ensure a strong bond of the deposit with the base metal, restrict their mixing and avoid the formation of cracks and other defects in the deposited layer.

Parts to be hard-faced are first preheated to 350°-500° C; the hard-faced parts are to be cooled slowly.

Hard-facing increases the service life of certain parts by 3 or 4 times, on the average, and enables worn parts to be repeatedly restored.

## Chapter 27

### QUALITY CONTROL AND REJECTS IN WELDING

#### 27-1. Deformation and Stresses in Welding

Considerable thermal stresses develop in welding during the heating and subsequent cooling of the work; residual stresses appear when the work has been completely cooled. At high stresses that exceed the yield point, the heated zone of metal is subject to plastic deformation.

Stresses and deformation in welding are the result of: (1) nonuniform heating and cooling of the metal; (2) shrinkage of the deposited metal in its transition from the liquid to the solid state, leading to tensile stresses which warp the work or form cracks in work of brittle metals, and (3) structural transformations in the deposited metal and in the heat-affected zone of the base metal which may be accompanied by changes in volume. Stresses, due to structural changes in the metal, are of importance only for hardenable steels (chiefly alloy steels) since the formation of martensite in hardening is associated with an increase in volume.

The magnitude of the deformation and stresses depends as well and to a great degree upon the shape of the work, its size and the character of the heating zone in welding. The stresses and deformation produced by the arc welding of work of simple shape is less marked. Gas welding causes increased deformation due to the larger extent of the heat-affected zone.

Welding stresses can be reduced or avoided by:

- (1) preheating the work before welding to reduce the temperature gradient and to retard cooling; the preheating temperature is determined by the properties of the metal;
- (2) annealing steel weldments at 550°-650° C;
- (3) lightly hammering multiple-layer welds to eliminate fine cracks in them.

The following measures can be recommended to reduce distortion in welding:

- (1) Use a back-step sequence to deposit the weld bead. This is a procedure in which a long weld is divided into 150 to 200 mm sections



welded separately in the direction opposite to the progress of welding the joint, thereby avoiding the concentration of heat in one place and reducing the heated area of the work.

(2) Before making the weld, deform the work by the same amount and in the direction opposite to that caused by welding. This procedure is usually resorted to for work with asymmetrically located welds.

(3) Balance the distortion, i. e., select a welding sequence in which each successive weld produces a distortion opposite to that produced by the preceding weld.

(4) Intensify heat dissipation from the work being welded. This reduces the volume of heated metal and, consequently, its distortion. Cooling is accomplished by immersing part of the work into water (only the place to be welded is on the surface) or by inserting copper pads under the work.

(5) Clamp the elements of various weldments rigidly in a special fixture. This method reduces distortion but increases internal stresses which may be relieved by subsequent annealing.

## **27-2. Weld Inspection**

Defects in the weld and the near-weld zones of the base metal reduce the mechanical strength of the joint to some degree.

The principal defects of welded joints include:

(1) poor fusion—the lack of coalescence between the deposited and base metal or incomplete penetration of the deposited metal into the base metal. Poor fusion is the result of faulty welding conditions and/or techniques;

(2) burning—oxidation of the metal in the weld and adjacent base metal. Burning is caused by a strong oxidising medium, an arc of excessive length, too slow motion of the heating facilities or excessively high welding current;

(3) burning holes—the melting of holes through the parts being joined by arc welding. The main causes are a high welding current, insufficient thickness of the metal or too small a root face;

(4) undercut—a groove melted into the base metal adjacent to the toe of the weld. The reasons for undercutting are nonuniform feed of the filler rod, improper position of the electrode or torch tip or excessive heating;

(5) overlap—protrusion of weld metal beyond the bond at the toe of the weld. Overlapping is due to improper welding conditions and speed;

(6) porosity—the formation of blowholes, gas pockets or roughness on the surface of the weld. This defect may be due to the presence of gases in the metal, moisture in the flux or coating, or rust on the welded edges or filler material;

(7) slag inclusions—the presence of nonmetallic substances in the metal. Slag inclusions are due to the contamination of the base and deposited metal by oxides, nonuniform melting of the electrode coating, high melting point and high viscosity of the slag or insufficient deoxidising of the metal in the weld;

(8) cracks in the weld may be due to heavy shrinkage and structural stresses in the metal (for example, the formation of martensite), extra-high amount of sulphur, phosphorus or carbon in the metal, or excessively rigid clamping of the parts being welded.

The following are the best known methods for detecting faults and defects in the quality control of weldments:

(1) Visual inspection of the weld and checking of its dimensions. This reveals such defects as undercuts, gas pockets, blowholes, cracks, porosity, unfilled craters, uneven welds and dimensional errors.

(2) Mechanical tests to determine the mechanical properties of the deposited metal and the welded joint. These tests establish the tensile strength, relative elongation, bending angle, impact strength and hardness. Mechanical tests may be conducted on the welded joint or only on the deposited metal. The shape and size of the test specimen are selected to comply with State Standards.

(3) Radiographic inspection of the weld, which is based on the different absorption of X-rays passing through the work by the metal and various nonmetallic substances. This method reveals porosity, blowholes, cracks, poor fusion and slag inclusions. The high penetrating capacity of  $\gamma$ -rays enables weldments with a thickness up to 300-350 mm to be inspected.

(4) Hydraulic (pressure) tests applied to equipment that is to operate under pressure.

(5) Air (pressure) tests to check the airtightness of the work.

(6) Ultrasonic inspection, based on the capacity of different media to reflect ultrasonic waves (with frequencies over 20,000 cps) in a different manner. Flaws in welds, such as nonmetallic inclusions, can be detected by this method in parts up to 5 m thick.

(7) Magnetic inspection methods, which utilise the dispersion of magnetic fluxes at the defects in the work. The magnetic flux is distorted so that lines of force are concentrated on the surface of the magnetised part at the edges of a discontinuity or other flaw. These lines of force attract finely divided magnetic particles which were previously applied uniformly to the whole surface of the work. The readily visible accumulation of these particles outlines the defect. This method reveals fine cracks and pores in the weld.

(8) Ammonia penetrant test. This leak test consists in filling a welded vessel with compressed air to which one per cent ammonia has been added and covering the welded joints with paper impregnated

ed with a 5 per cent solution of mercuric nitrate. Black spots appear on the paper in case of leakage.

(9) Fluorescent penetrant inspection. The part being tested by this method is immersed for 20-30 minutes in a mixture of kerosene and oil. Then it is wiped dry and immersed in magnesia powder which adheres at the places where the oil appears (over the cracks). This method is applied to reveal fine cracks.

(10) Metallographic inspection. Such inspection reveals the macro- and microstructure of the metal, as well as cracks, blowholes, poor fusion, burning, overheating, the presence of nitrides and other defects of the weld.

## *Chapter 28*

### ARC AND OXYGEN CUTTING OF METALS AND ALLOYS

Ferrous and nonferrous metals and their alloys can be cut either by an electric arc or by a gas flame.

#### **28-1. Arc Cutting**

Both carbon and metal electrodes are employed in the arc cutting of metals.

In these processes the severing or removing of metals is effected by melting the metal with the heat of an electric arc at the cut. The application of compressed air to blow out the molten metal speeds up the cutting process.

This method is used in the demolition of steel structures and pipeline mains, for cutting up scrap, removing the gating system from castings, cutting nonferrous metals, steel and cast iron, burning through holes and also for repair and assembly operations.

Straight polarity is commonly applied for carbon-arc cutting. The electrodes are from 10 to 20 mm in diameter and the current may vary from 400 to 800 A depending upon the depth of the cut.

The use of metal electrodes improves the quality of the cut, reduces the width of the kerf and produces more evenly cut edges.

#### **28-2. Oxygen Cutting**

In various gas cutting processes the temperature necessary for the cutting process is maintained by means of a flame obtained from the combustion of a fuel gas.

The two most widely used processes of this type are straight oxy-gas cutting and the metal powder and chemical flux cutting processes.

*Oxy-gas cutting* is based upon the ability of certain metals to burn in oxygen with the evolution of a great deal of heat thereby melting the metal and forming oxides.

This method is suitable for cutting only metals in which the ignition temperature is lower than the melting point and the melting point of the formed oxides is lower than that of the metal itself. Moreover, the oxides must have fair fluidity. The heat conductivity of the metal must be low so as to concentrate the heat.

Carbon steels with a carbon content up to 0.7 per cent and low-alloy steels are cut by this technique. High-alloy steels are preheated to 650°-700° C before cutting. Metals which cannot be effectively cut by this method are: cast iron, since its melting point is equal to 1200° C while its ignition temperature is 1350° C; high-alloy chromium and chrome-nickel steels; and nonferrous alloys since the melting point of their oxides is higher than that of the base metals.

The so-called *metal powder cutting process* consists in introducing into the cutting zone a finely divided powder which is carried from a hopper by the stream of oxygen while the oxy-acetylene cutting torch travels along the reaction zone. The powder burns in the oxygen stream and generates additional heat which facilitates the melting of the refractory oxides and their removal from the cutting zone by the stream of oxygen. This process has proved suitable for cutting stainless and heat-resistant steels, cast iron and certain nonferrous metals.

The powders employed consist mainly of iron fillings with particles up to 0.2 mm in size. They also contain a small amount of ferrophosphorus, aluminium and other components.

Cutting is frequently employed in place of roughing and planing operations on machine tools to remove extra metal from the surface of parts and to eliminate various defects on steel ingots, castings and rolled stock by what is called scarfing.

Oxy-gas cutting is performed by means of ordinary gas-welding equipment except that the welding torch is replaced by a cutting torch which delivers the gas mixture for preheating and oxygen for burning the metal. The cutting torch has interchangeable nozzles with preheating (external) and cutting (internal) orifices.

The mechanisation of cutting processes enables more uniform motion of the torch to be effected, improves the quality of the cut and substantially increases the cutting speed.

In semiautomatic oxygen cutting machines the torch is traversed by power but is guided by hand with the aid of various devices (tracks or guide bars, templates, disks, etc.). Such machines are widely used in industry and construction.

In automatic machines, not only is cutting torch travel mechanised, but the direction of the cut is controlled automatically. Stationary shape cutting machines can make straight or irregular cuts to produce the required shape. They incorporate mechanisms which either provide longitudinal and transverse movements of the working units or are designed with a jointed swivelling linkage. The cutting torches of these machines reproduce the movement of a magnetic tracing roller which follows the contour of a template, thus cutting out a part of a shape sufficiently accurate to eliminate subsequent machining in many cases.

On certain special machines cutting is performed to the contour of a template by a magnetic tracer head and to a drawing or layout lines by means of a hand-guided head.

Machines have been developed with photoelectric tracing devices which follow the lines of a drawing; machines for cutting sheet metal; remote-controlled machines, and machines in which a pantograph in the tracing device enlarges or reduces the size of the shape cut out in relation to the template.

Not only acetylene is used for cutting metals. Methane, petroleum gas, lighting gas, coke-oven gas and others are also extensively employed.

Special kerosene and gasoline cutting torches are used for cutting with liquid fuels.

Certain cutting torches of special design can cut steel up to 2 m thick.

A procedure used to increase the rate of production in cutting sheets from 2 to 10 mm thick consists in piling them into stacks up to 150 mm high and cutting the whole stack simultaneously by a suitable manually controlled or automatic method.

A combined process, known as oxygen-arc cutting, involves the preheating of the metal to be cut by an electric arc. The electrode is a steel tube through which the cutting oxygen is delivered.

Techniques have come into use recently for cutting metal under water in ship building and repair work. The steel electrode used in underwater arc cutting has a special waterproof coating.

Underwater oxygen cutting can be accomplished with an oxy-hydrogen torch which applies to the cut, not only a powerful preheating flame, but compressed air as well for forcing away the water from the flame.

The construction of the jointed-type shape-cutting machine, model ACIII-2, is shown in Fig. 177a. It is designed for the production of identical parts in large lots to a template whose contour is followed by a magnetic tracing roller. The hinged arms 2 and 3 swivel easily on ball-bearing joints and are mounted on the heavy column 1. Motor 7 and the tracing head with magnetic coil 5 are mounted at

the upper part of external arm 3. A steel pin with a knurled or serrated end (the magnetic roller) rotates inside the coil. When current passes through the winding of the coil the pin is magnetised, the roller is attracted to the edge of steel template 6 and rolls along its contour, describing the shape to be cut out. Templates are secured on bars 8 which are fastened to overarm 9. Cutting torch 4, mounted in the lower part of hinged arm 3, moves along the sheet being

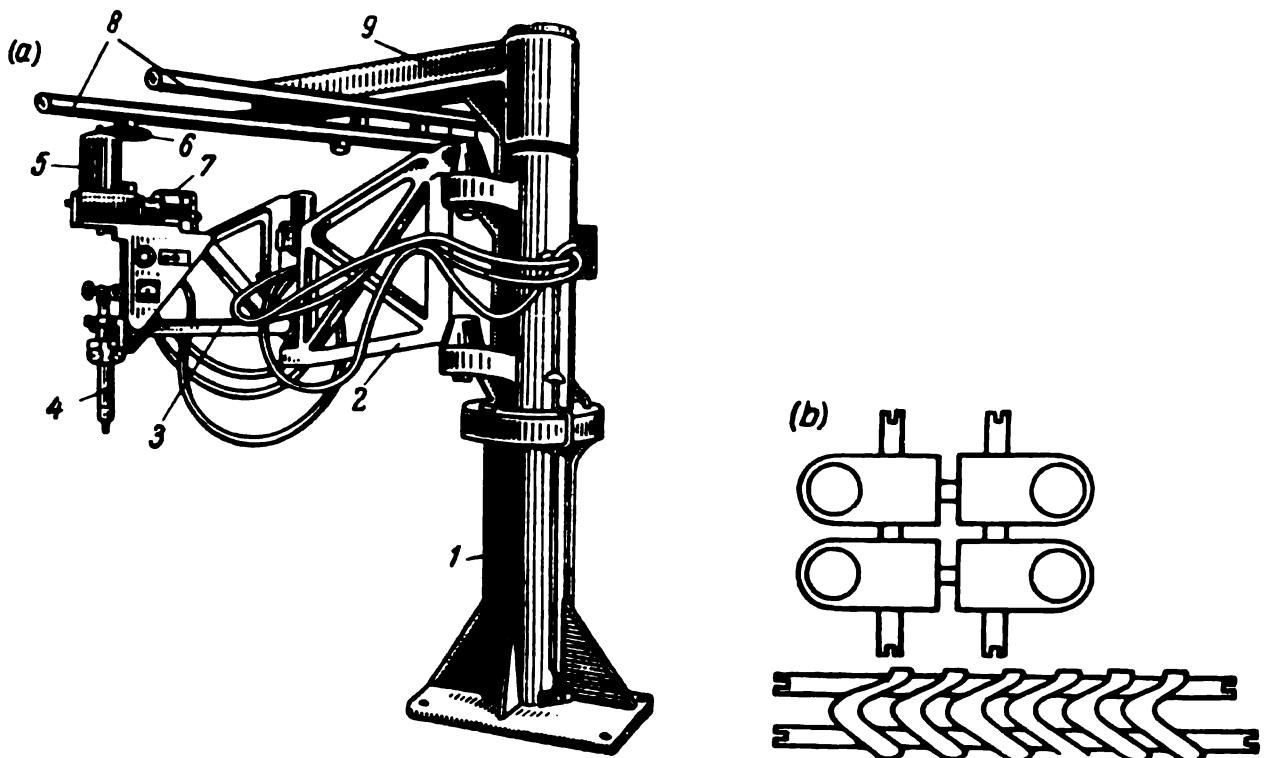


Fig. 177 Precision oxygen shape-cutting machine, model ACIII-2:

(a) general view; (b) composite templates

cut describing a contour whose shape and size correspond to those of template 6. The stream of oxygen cuts this shape out of a sheet or plate lying on a table in front of the machine.

Model ACIII-2 can be used to cut parts of any shape with overall dimensions of  $750 \times 1500$  mm from sheet or plate up to 100 mm thick. The cutting accuracy is very high—within 0.3-0.5 mm.

Composite templates are being used at present in many Soviet plants on the ACIII-2 machines. They speed up the cutting operation and reduce the amount of scrap in cutting out small similar parts since the necessity of shifting the template or sheet for each new part is excluded. Examples of such templates are illustrated in Fig. 177b.

## Chapter 29

### BRAZING AND SOLDERING OF METALS AND ALLOYS

#### 29-1. Types of Brazing and Soldering

*Brazing and soldering* are processes for joining solid metal components by introducing between them a molten alloy (brazing filler metal or solder, as the case may be) having a lower melting point than that of the base metal of the components being joined.

To obtain a strong joint it is necessary that the brazing alloy or solder should properly wet the surfaces of the base metals and form solutions with them. The greater the degree of interdiffusion between the molten alloy, or solder, and the base metals the higher the mechanical strength of the joint will be. Besides this, the joint strength depends upon the cleanliness of the joined parts in the area of the joint; therefore, a precleaning operation is required.

Brazing and soldering can be employed to join parts of carbon and alloy steels of all grades, nonferrous metals and their alloys, as well as dissimilar metals and alloys.

The distinction between brazing and soldering is in the temperature range in which the process is performed. This, in turn, depends upon the melting point of the filler metal used. Solders employed in soldering operations have a low melting point (up to 400°C) and the junction has a relatively low mechanical strength. Brazing alloys, on the other hand, have a much higher melting point (up to 900°C) and produce a joint with high mechanical strength.

Soldering is employed when liquidtight joints are required, having a comparatively low mechanical strength (the tensile strength of a soldered seam ranges from 3 to 10 kg per sq cm). The joints obtained by brazing have a much higher tensile strength (from 31 to 44 kg per sq cm).

*Solders* are tin-, lead-, cadmium-, bismuth- or zinc-base alloys. They are often referred to as soft solders to distinguish them from silver (hard) solders used in one brazing process. The Soviet grades of solder, OC-90, OC-40, OC-18, etc., contain, respectively, 90, 40 and 18% Sn, and from 0.15 to 2% S (remainder—Zn).

*Brazing filler metals* are alloys based on copper, silver, aluminium, magnesium and nickel. Grades ПМЦ-54 and ПМЦ-36 contain 54 and 36% Cu, respectively (remainder—Zn), while grades ПСР-12 and ПСР-45 contain 12 and 45% Ag, and 36 and 30% Cu, respectively (remainder—Zn).

## 29-2. Brazing and Soldering Techniques

The soldering process comprises the preparation of the surfaces for soldering and the actual soldering operation.

- *Surface preparation* involves fitting the surfaces to each other, cleaning them mechanically of dirt, grease and oxide films, and covering the cleaned surfaces with a flux.

Fluxes are applied to protect the cleaned surfaces against oxidation during heating and soldering, and also to improve the wetting of the base metal with the molten solder. The fluxes commonly used in soldering are zinc chloride, mixtures of zinc chloride and ammonium chloride (sal ammoniac), and rosin. Fluxes in paste form, containing zinc chloride, sal ammoniac, rosin, animal fat or castor oil, and water, are sometimes used.

After completing surface preparation, soldering proper is done.

Metals may be soldered by means of a hand-type soldering iron, a blowtorch, by dipping in molten solder and other methods.

A soldering iron is made of copper. The tip of the iron is tinned before applying it to the work.

Soldering irons may be of the plain, gasoline, gas or electric types.

To do a soldering job the iron is heated to 250°-300°C, the tip is dipped into the flux, tinned with the solder and then applied to the parts prepared for soldering to heat them at the joint. At the same time, solder is applied to the joint where it is melted by the iron moving along the joint and enters the clearance. Here the solder cools and forms the seam.

In brazing, the parts are first precleaned and a flux is applied to them. Then the parts are heated together with the filler metal to the melting point of the latter.

In certain cases the brazing and soldering of metals lend themselves well to the use of mechanised and automatic equipment. Mechanised and automatic brazing methods include resistance brazing, induction brazing, arc brazing, dip brazing and others.

The selection of a mechanised brazing method should be based on the scale of production, type of filler metal, work material and the size of the joined components.

Induction heating of the metal in brazing or soldering by a high-frequency current deserves special attention. This method is based on the heating of the components to be joined by means of an alternating current induced in the metal when it is placed in an alternating magnetic field with rapidly reversing lines of force.



Special heating coils, called inductors, are employed in high-frequency induction heating. These inductors are made of copper tubing and are of a shape corresponding to that of the work.

In mass production, use may be made of automatic induction brazing (or soldering) on a conveyer belt. After surface preparation and assembly on the belt the work passes through the inductor which heats it to the required temperature by automatic current control. The filler metal melts and brazing or soldering is accomplished.

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## THE MACHINING OF METALS

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### *Chapter 30*

#### FUNDAMENTALS OF MACHINING ACCURACY

##### **30-1. Interchangeability, Nominal Size and Limits of Size**

In modern mechanical engineering a great many types of machines are manufactured by flow-type mass production methods in which the separate units are assembled, as is the machine itself, without the need for individual fitting operations performed on the component parts or units.

The possibility of replacing parts or units of a machine during assembly or subsequent repairs without resorting to fitting operations is called *interchangeability*. Interchangeable assembly manufacturing is achieved by establishing the permissible degree of inaccuracy in making the parts of a machine, i. e., by establishing the amount of permissible deviation of the actual dimensions from the nominal ones.

*The nominal sizes* are the dimensions of a machine part specified in the part drawing. They are assigned on the basis of design considerations, theoretical strength calculations or experimental data and are suitably rounded off to whole-millimetre values. This rounding off of nominal sizes is of immense economical importance since it limits their diversity and total number to a definite range of standard dimensions.

It is very difficult to obtain the assigned nominal size in machining a metallic workpiece on a machine tool because of definite and inevitable errors of the machine tool, elastic strain of the holding fixture and the cutting tool, variations in the temperatures of the workpiece and the tool, differences in the skill of various operators and many other factors. Moreover, the maximum possible size accu-

racy is not always required. Therefore, to ensure parts interchangeability, two limits of size are assigned—maximum and minimum—within which the dimension must be maintained.

### 30-2. Tolerances and Machining Allowances

The difference between the maximum and minimum limits of size of a part, i. e., the total permissible variation of a size, is called *the tolerance*. Tolerances may have upper and lower deviations.

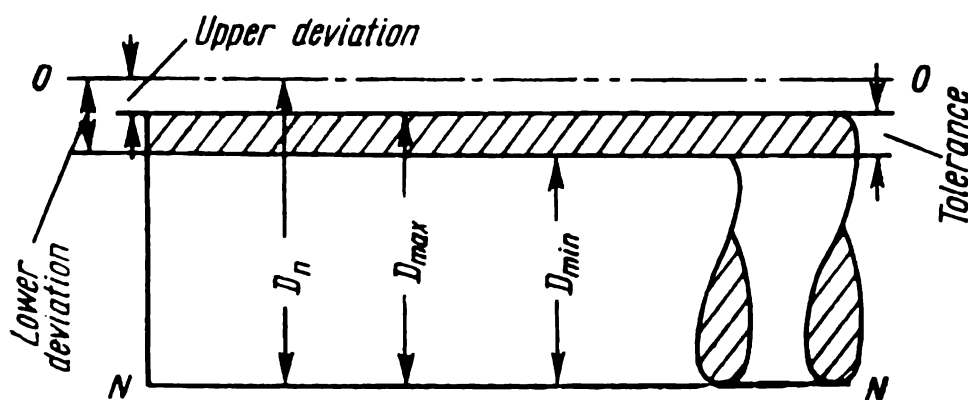


Fig. 178. Tolerance and the tolerance zone

The nominal size  $D_n$  serves as the origin from which the deviations to either side are specified and is usually designated by a zero line 0-0. In Fig. 178 the deviations are specified from the nominal size in the minus direction.

The difference between the maximum limit of size (or simply maximum limit) and the nominal size is called *the upper deviation*; that between the minimum limit and the nominal size is *the lower deviation*. Thus, the actual size  $D$  determined by measurement is somewhere between the maximum  $D_{max}$  and minimum  $D_{min}$  limits and in particular cases may be equal to one of them. This can be expressed by the inequality  $D_{max} \geq D \geq D_{min}$ .

On drawings, tolerances are designated by putting down the nominal size which is followed by the maximum deviations. The latter characterise the tolerance zone and are written, as a rule, with the upper deviation arranged above the lower one as, for instance,  $50_{-0.05}^{+0.02}$ .

All dimensions of the part that are within the tolerance zone  $D_{max} - D_{min}$  are acceptable. Such parts are considered to have passed inspection; otherwise they are rejected.

The selection of a rational tolerance must be based on the nominal size of the part, the tolerance system being used and the chosen grade of accuracy since the same tolerance characterises different degrees of accuracy when it is assigned to different nominal sizes. For example, a tolerance of 0.1 mm is wide for a dimension of 4 mm but is narrow

for a dimension of 400 mm. Tables of standards are employed to select rational tolerances in accordance with the given nominal size.

To obtain a finished part of specified shape, size, accuracy and surface finish, the blank is taken with a definite amount of surplus metal (stock) which is subsequently removed in the process of machining. This surplus stock is called *the machining allowance*. In actual practice measures are taken to maintain the minimum feasible machining allowances thereby reducing both the machining cost and the metal consumption per unit of the finished product.

### 30-3. Shaft Basis and Hole Basis Tolerance Systems

Dimensions in the engineering industries are classified as mating and free dimensions.

An example of mating dimensions is the outside diameter of the piston in a pneumatic power forging hammer and the bore of the cylinder in which the piston reciprocates. In this case the cylindrical surface of the piston is a typical shaft surface while that of the bore is a typical hole surface.

For the sake of brevity any external, enveloped surface of mating parts is called a *shaft* and any internal, enveloping surface is called a *hole*. This may refer to surfaces whose shape in no way resembles a cylinder; for instance, in a key and keyway fit the key is regarded to be the shaft and the keyway—the hole.

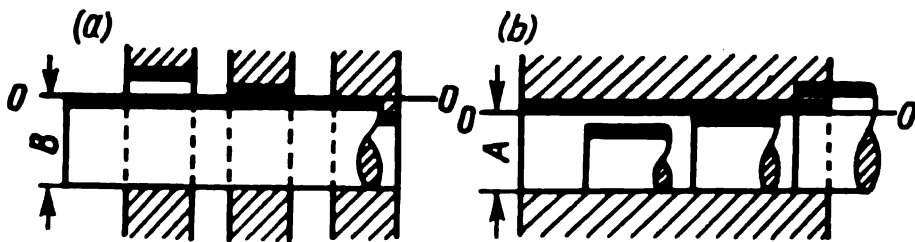


Fig. 179. Tolerance systems:  
(a) shaft basis system; (b) hole basis system

*Free dimensions* specify the size of nonfunctional surfaces of a part. Examples are the length of the container sleeve in a hydraulic press, outside diameter of a flange, diameter of a rivet head, etc.

A system of tolerances in which the limits of size of the shaft are constant for all fits is called *the shaft basis system* (Fig. 179a). The various types of fits are obtained by suitably varying the tolerance zone of the mating hole. On drawings the basic shaft is designated by the letter B with a subscript specifying the grade of accuracy. The letter follows the nominal size, for example—50B<sub>8</sub>. In this system the actual size of a shaft that is within the tolerance limits

is always less than the nominal size. As a particular case it can equal the nominal size but can never be larger.

*The hole basis system* (Fig. 179b), on the other hand, is one in which the limits of size of the hole are constant for all fits. Here the various fits are obtained by varying the tolerances of the mating shaft. The symbol designating the basic hole is the letter A followed as before by the subscript indicating the grade of accuracy. The letter symbol follows the nominal size, as for example 50A<sub>3</sub>. In this system the actual size of a hole that is within the tolerance limits is always more than the nominal size; it can equal the nominal size as a particular case but can never be less.

### 30-4. Fits

When parts are assembled into units and units into the finished machine, mating surfaces of similar shape are jointed together. Either one of them fits into the other or they abut together. The character of the joint, or association, obtained depends upon *the fit* by which is meant the degree of tightness or freedom for relative movement between the mating parts. The fit is the result of the difference in the sizes of the mating parts.

Fits are of three general types: clearance, transition and interference.

In a *clearance fit* the mating parts can move in relation to each other. This type of fit is employed, for instance, in the association between a piston and its cylinder. To obtain a clearance fit it will be necessary that the cylinder bore be somewhat larger than the diameter of the piston. This difference between cylinder and piston diameters (and, in the general case, between hole and shaft diameters) is called *the clearance*. If, for example, the hole diameter is 50 mm and the shaft diameter is 49.8 mm, the clearance is 0.2 mm. It follows that the clearance is always a positive value.

In an *interference fit* the mating parts are joined tightly together; no relative motion is possible. This condition is achieved by making the shaft somewhat larger than the hole. This negative clearance between the mating parts is called *the interference*.

*Transition fits* have their limits of size so prescribed that either a clearance or an interference may result when the mating parts are assembled.

It has been found in practice that a number of different fits of each type are required which can provide different degrees of tightness or freedom between mating parts. In Soviet engineering the most extensively employed fits of the clearance type are: (1) slide fit (S), (2) easy slide fit (Se), (3) running fit (R), (4) slack running fit (Rs) and (5) loose running fit (R1). Those of the interference type are:

(1) shrink fit (Sh), (2) heavy drive (Dh) and (3) light drive fit (Dl). The four transition fits are: (1) force fit (F), (2) tight fit (T), (3) wringing fit (W) and (4) push fit (P).\*

Slide fits have a very small clearance and are applied when the mating parts move slowly in reference to each other (for instance, the tailstock spindle in a lathe tailstock, the feed movement of the spindle quill in a drilling machine, etc.). An easy slide fit also has a small clearance and serves to ensure alignment between the shaft and hole (spindles of lathes, dividing heads, etc.). Running fits have appreciable clearance and are used in engineering for rotation at moderate speeds (crankshafts in their main bearings, throttles in the valve sleeves of a steam-and-air power forging hammer, etc.). A slack running fit has considerable clearance. It may be employed for rotation at high speeds or for moderate speeds if the bearings are very long (shafts of centrifugal pumps, shafts in the drives of cylindrical grinding machines, etc.). The largest clearances are found in loose running fits. Such fits are employed for rotation at very high speeds and if misalignment of the mating parts may occur in assembly (shafts in especially long bearings, idle pulleys on their shafts, etc.)

Only interferences are found in shrink, heavy drive and light drive fits which are used for fixed permanent (inseparable) joints in which no additional fixing elements are required. Elastic strain developed on the mating surfaces in the process of assembly prevent relative displacement of the component parts. Examples are the steel tyres of railway car wheels, gears on the intermediate shafts of trucks, bushing in the gear of a lathe headstock, etc.

Transition fits ensure proper location of shafts in holes; either clearance or interference may occur between the mating parts. Therefore, fixed joints with transition fits require additional fastening elements (keys, cotters, pins, etc.) to prevent relative motion of the parts. The difference between the sizes of the shaft and hole are very small as is, consequently, the clearance or interference.

\* We have used as fit symbols the letters which correspond to the English names of the fits. It must be understood, however, that in Soviet engineering drawings fits are specified by Russian capital letters which correspond, in the same manner, to the Russian names of the fits. These symbols are:

Shrink (Sh)	Гр	Push (P)	П
Heavy drive (Dh)	Пр	Slide (S)	С
Light drive (Dl)	Пл	Easy slide (Se)	Д
Force (F)	Г	Running (R)	Х
Tight (T)	Т	Slack running (Rs)	Л
Wringing (W)	Н	Loose running (Rl)	Ш

English names, given here to the fits in the U.S.S.R. standard, have been selected to broadly characterise their properties. It is not to be inferred that they coincide in tolerances with fits of the same name employed in Great Britain or the United States.—*Tr.*

Force fits are applied for mating parts that are not to be disassembled during their whole term of service (gears on the shafts of a concrete mixer, forging machine, etc.). Tight fits provide less interference than force fits and are employed for parts and units that may be replaced in overhauling the machine (stepped pulleys on the drive shafts of a conveyer, cylindrical grinding machine, etc.). A wringing fit provides either zero interference or a clearance and is met with in parts and units which can be replaced without difficulty during minor repairs (gears of machine tools). A push fit is characterised by a clearance and is employed for units that must be disassembled during operation of a machine (change gears, slip bushings, etc.).

The class of fit is usually designated in drawings by the letter symbol followed by a subscript indicating the specified grade of accuracy. For example, a slack running fit in the 4th grade of accuracy is denoted by  $Rs_4$ .

### 30-5. Manufacturing Accuracy and Surface Finish

**Manufacturing accuracy.** The principle of interchangeable assembly manufacturing demands that the degree of accuracy with which any part is made should be in agreement with its function in the finished machine and the operating conditions. In the engineering industries, *the manufacturing accuracy* is the degree by which the actual size and geometrical shape of a part conforms to the nominal size and shape specified in the part drawing. Higher accuracy than required by functional considerations leads to an increase in machining costs, reduction in output of the machine tools, etc.

The manufacturing accuracy of a part is usually specified by two limiting values—the maximum and minimum permissible sizes whose numerical value depends upon the fit desired and the size of the part.

Ten grades of accuracy are applied in Soviet machine building: 1, 2, 2a, 3, 3a, 4, 5, 7, 8, and 9. There is no 6th grade since, in accordance with the International Tolerance System (ISA) the interval between the 4th and 7th grades was to be divided into three ranges.

Each grade of accuracy is characterised by its own tolerances for shafts and holes.

The highest grade of accuracy is the 1st; it can be obtained in machining by precise grinding, precision turning and boring, sizing and lapping. This grade finds application in instrument and precision machine tool manufacture, in making the components of antifriction bearings, etc.

The 2nd and 2a grades are achieved by accurate finish grinding, accurate broaching and finish reaming; they are used in the manu-

facture of precise machinery, machine tools, automobile and aircraft engines, electric motors, pneumatic equipment and other similar items.

The 3rd and 3a grades of accuracy require rough grinding or finish turning and are employed in industries manufacturing general-purpose machinery, automobiles, tractors, railway cars, Diesel engines, steam engines and turbines.

The 4th grade of accuracy may be obtained in ordinary turning and accurate drilling and core-drilling. It finds applications in Diesel engine locomotives and agricultural machinery.

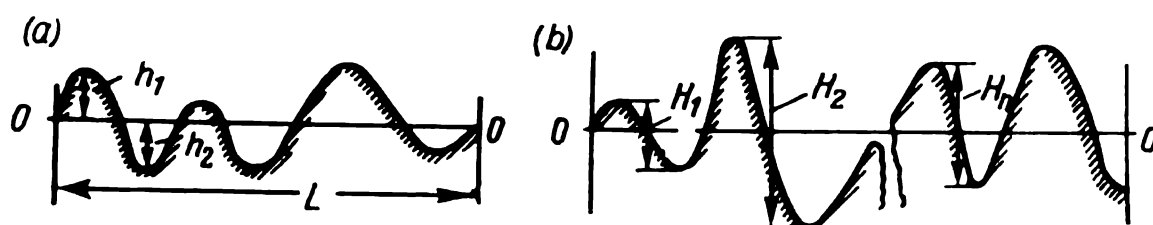


Fig. 180. Profiles of the microirregularities on a surface:

- (a) root mean square average deviation of the microirregularities from a mean line;  
 (b) arithmetical average height of microirregularities

The 5th grade may be the result of rough turning and drilling and is used in manufacture of components of agricultural machinery where no high requirements are made as to the fitting of mating parts.

The 7th, 8th and 9th grades are the accuracy obtained in casting, forging and rolling of metals or in exceptionally rough machining. Tolerances in these grades are assigned to the free dimensions of various parts and to blanks having allowances for subsequent machining operations.

**Surface finish.** Irregularities produced on the surface of manufactured parts by the action of the cutting tool in machining have the form of ridges and valleys which are visible to the naked eye after rough machining and can be seen through a microscope after finishing operations. These ridges and valleys are called the microirregularities of the surface.

The surface quality of an article depends upon the height (peak-to-valley distance) of the microirregularities.

An idea of the profile of the microirregularities can be obtained by properly magnifying a cross section of the surface at a place where the ridges and valleys are evident (Fig. 180).

Two criteria are used to evaluate the surface finish of a part. One is the root mean square average deviation  $H_{rms}$  of the microirregularities from a mean line and the other is the arithmetical average height  $H_{aa}$  (peak-to-valley distance) measured over a short distance on the surface (usually from 1 to 3 mm). The  $H_{rms}$  value is determined on the microprofile of the surface on the basis of the deviations from



a mean line, or centre line,  $0-0$  (Fig. 180a). This line is parallel to the theoretical surface and is drawn in such a manner that the sums of the areas contained between it and those parts of the profile that lie on either side of it are equal. Then the root mean square average is defined numerically as follows:

$$H_{rms} = \sqrt{\frac{1}{L} \int_0^L h^2 dL}, \quad (91)$$

where  $L$  = length of the part of the surface over which the average is taken,

$h$  = height of peaks or depth of valleys at increments into which the length  $L$  is divided.

For simplicity in practice an approximate value may be obtained by replacing integration with summation of the heights at  $n$  points equally spaced. Then

$$H_{rms} = \sqrt{\frac{1}{n} (h_1^2 + h_2^2 + h_3^2 + \dots + h_n^2)}. \quad (92)$$

The arithmetical average deviation in the height of the microirregularities, or the average height of microirregularities,  $H_{aa}$  (Fig. 180b) is determined by the formula:

$$H_{aa} = \frac{1}{n} (H_1 + H_2 + H_3 + \dots + H_n), \quad (93)$$

where  $H$  = peak-to-valley height of the microirregularities,  
 $n$  = number of ridges in the length over which the average is taken.

Values of  $H$  that are considerably outside of the range being considered are neglected.

The numerical characteristics of microirregularities are specified in a U.S.S.R. standard which establishes two scales of surface finish: *general*—comprising groups of finish and *subdivided*—in which the groups are further divided into classes. A third scale with series of finish has been established for certain industries, for instance, engine manufacture, but it is seldom applied.

The first (general) scale of surface finish consists of four groups. Group I is obtained in roughing blanks, in permanent-mould casting and in die forging. The work surface shows prominent traces of the processing operation. Group II is the result of ordinary turning, finish milling, rough grinding, hand filing and scraping. Here the traces of machining can be seen with the naked eye. Group III refers to finishes produced in finish turning, finish grinding and pol-

ishing; no traces of machining are seen with the naked eye. Finishes of the highest quality, group IV, are obtainable in finishing operations such as buffing and lapping, and the surface is mirror smooth.

The second (subdivided) scale of surface finish comprises 14 classes in which the 1st class refers to the roughest surfaces and the 14th class—to the finest surfaces.

A proper selection of surface finish quality is of great economical importance. There is less friction between mating parts having good surface finishes; they wear more slowly, have a longer service life and are less susceptible to corrosion.

## Chapter 31

### FUNDAMENTALS OF METAL CUTTING. TYPES OF METAL-CUTTING MACHINE TOOLS

#### 31-1. Machining Methods. Principal Elements of Metal Cutting

**Machining methods.** Metal-cutting processes are performed on machine tools by means of various types of cutting tools.

In the machining process, working motion is imparted to the work and cutting tool by the mechanisms of the machine tool so that the work and tool travel relative to each other.

A combination of two types of working motion are required in machining: (1) *primary cutting motion* at a speed higher than other motions and (2) *feed motion* at a speed less than the primary cutting motion.

The most widely applied metal-cutting machining methods are turning, drilling, milling, planing and grinding.

*Turning* is performed on machine tools of the lathe type. Here the primary cutting motion  $v$  (rotary in this case) is imparted to the work and the feed motion  $s$  (along the axis of the work) is imparted to the tool which in this case is a single-point lathe tool (Fig. 181a).

*Drilling* is done on drilling machines of many types; both the primary cutting motion  $v$  (rotary) and the feed motion  $s$  (rectilinear) are imparted to the cutting tool—a drill (Fig. 181b).

In *milling* on milling machines the cutting tool, called a milling cutter, provides the primary cutting motion  $v$  (rotary) while the feed motion  $s$  (rectilinear) is imparted to the work (Fig. 181c).

The like operations, *planing and shaping*, are performed on planers and shapers. In shaping, the primary cutting motion  $v$  (reciprocating) is imparted to a single-point tool and the feed motion  $s$  (rectilinear and intermittent) to the work (Fig. 181d). In planing, the opposite is true insofar as the working motions are concerned (Fig. 181e).

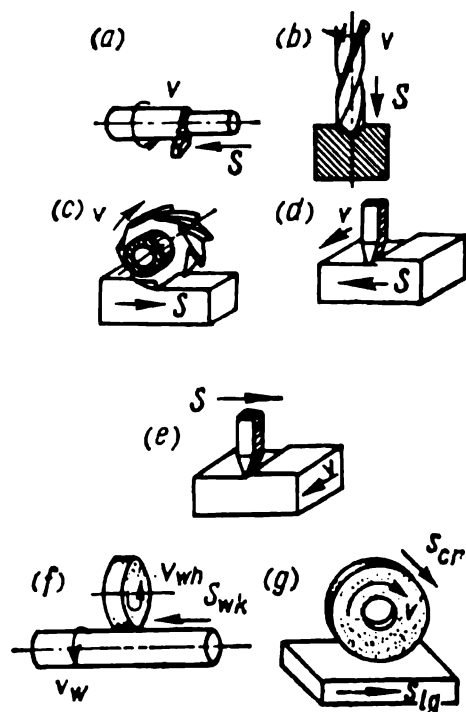


Fig. 181. Principal machining methods:

(a) turning; (b) drilling; (c) milling; (d) shaping; (e) planing; (f) cylindrical grinding; (g) surface grinding

*Grinding* is performed by a great variety of types of grinding machines. In all cases, however, the primary cutting motion  $v$  is rotary and is imparted to the grinding wheel. The feed motion  $s$  (rotary and reciprocating) is imparted to the work in external cylindrical grinding (Fig. 181f). In surface grinding a longitudinal reciprocating feed motion is imparted to the work and a cross-feed motion  $s$ —to the grinding wheel (Fig. 181g).

In all these machining methods the speed of the primary cutting motion  $v$  is many times higher than the rate of feed  $s$ .

**Principal elements of metal cutting.** Cutting conditions are characterised, in the main, by such elements as: (1) cutting speed, (2) feed, (3) depth of cut, (4) undeformed-chip cross section (width and thickness of the undeformed chip) and (5) piece, or cycle, time and machining time.

*The cutting speed* is the distance travelled by the work surface in a unit of time in reference to the cutting edge of the tool.

The cutting speed  $v$  is determined by the formula

$$v = \frac{\pi D n}{1000} \text{ m per min,} \quad (94)$$

where  $D$  = diameter of the work surface, in mm,  
 $n$  = workpiece speed, in rpm.

*The feed  $s$*  is the movement of the tool cutting edge per revolution of the work (Fig. 182); in turning it is expressed in millimetres per revolution (mm/rev).

*The depth of cut  $t$*  is measured in a direction perpendicular to the work axis and, in straight turning in one pass, it is found from the formula:

$$t = \frac{D - d}{2} \text{ mm,} \quad (95)$$

where  $D$  = diameter of the work surface, in mm,  
 $d$  = diameter of the machined surface, in mm.

The undeformed-chip cross section  $f$  is the cross-sectional area of the chip before it is cut from the work and is equal to the product of the feed multiplied by the depth of cut:

$$f = st \text{ sq mm.} \quad (96)$$

The time required to machine one workpiece is called the *cycle time*; it includes the machining time and the handling time.

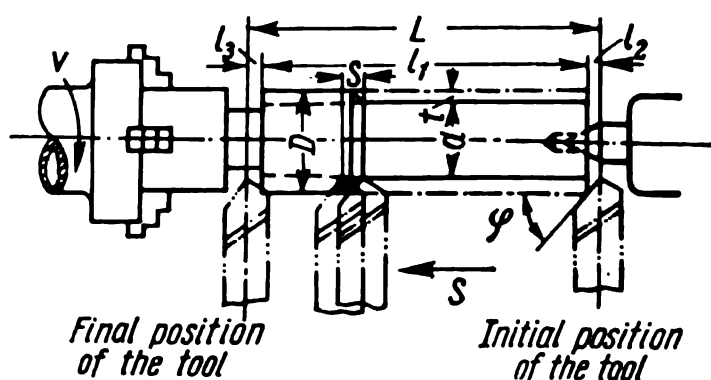


Fig. 182. Elements of metal cutting

The *machining (or cutting) time* is that in which actual metal cutting is being performed. The machining time required for turning a single workpiece in several passes (cuts) is determined by

$$T_m = \frac{Li}{sn} \text{ min,} \quad (97)$$

where  $L$  = length of tool travel in the feed direction, in mm,  
 $i$  = number of passes,  
 $s$  = tool feed, in mm/rev,  
 $n$  = speed of the work, in rpm.

The length of tool travel  $L$  in straight turning is the sum of work surface length  $l_1$ , tool approach length  $l_2$  and the tool overtravel  $l_3$ , so that

$$L = l_1 + l_2 + l_3 \text{ mm.} \quad (98)$$

The tool approach length depends upon the depth of cut and plan approach angle  $\phi$ ; it is determined from the formula

$$l_2 = t \cot \phi. \quad (99)$$

Overtravel  $l_3$  is required to avoid the formation of a burr at the end of the cut and is taken from 1 to 3 mm depending upon the work diameter.

The *handling time* is the time required for the operator to do various auxiliary operations such as loading and unloading the workpiece, starting and stopping the machine, and advancing and with-

drawing the tool. These are operations repeated in each cycle and without which machining could not be carried out. The time spent in changing dull tools, chip removal, machine lubrication, etc., is called servicing time.

The standard time (per piece) is determined by the formula:

$$T_p = T_m + T_h + T_s + T_f \text{ min,} \quad (100)$$

where  $T_m$  = machining time, in min,

$T_h$  = handling time, in min,

$T_s$  = servicing time, in min,

$T_f$  = time required for rest and personal needs (called the fatigue allowance), in min.

If the standard time per piece is known there will be no difficulty in determining *the output, or rate of production*, of the machine tool which is the number of workpieces machined in a unit of time, for instance, in an hour. The hourly output is

$$A = \frac{60}{T_p} = \frac{60}{T_m + T_h + T_s + T_f} \text{ pcs per hr.} \quad (101)$$

It is evident from formula (101) that the rate of production can be increased by reducing the machining, handling and servicing times.

Machining time can be reduced substantially by employing the most expedient speeds and feeds, and by introducing new and progressive processing methods. Mechanisation of manual operations, automatised handling, application of efficient fixtures and other work holding devices and proper training of operators in using time and labour saving methods are all measures that can reduce the handling and servicing times.

### 31-2. Elements and Angles of Single-point Tools

The single-point tool is the most widely used cutting tool.

A **single-point tool** consists of the point *A* and the shank *B* which is clamped in a tool post (Fig. 183) The point is bounded by the face 1 (on which the chips impinge as they are cut from the work), side flank 2, end flank 3 and the base 4. The side-cutting edge *a-b* is formed by the intersection of the face and side flank; the end-cutting edge *a-c*—by the intersection of the face and end flank. The conjunction *a* of the end- and side-cutting edges is called the nose. The side-cutting edge is the element that cuts the chip from the work.

Three types of surfaces are defined in machining: (1) *the work surface*, from which the chip is cut (Fig. 184), (2) *machined surface* formed or generated after removing the chip and (3) *the cutting surface* which is formed by the side-cutting edge of the tool.

The references from which the tool angles are specified are *the cutting plane* and *the basic plane* (Fig. 184).

The first of these is a plane tangent to the cutting surface and passing through the side-cutting edge while the second is a plane par-

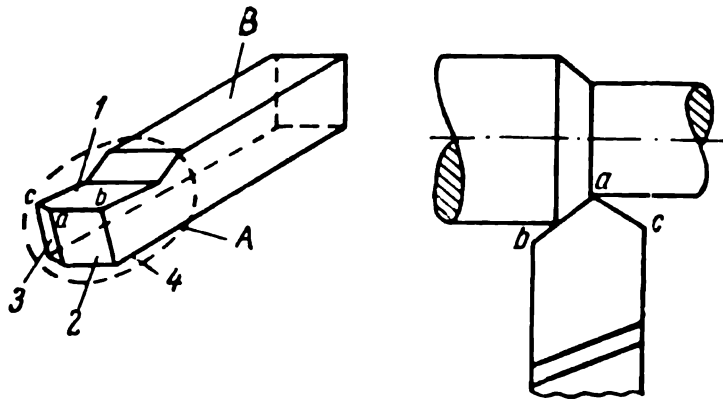


Fig. 183. Elements of a single-point tool

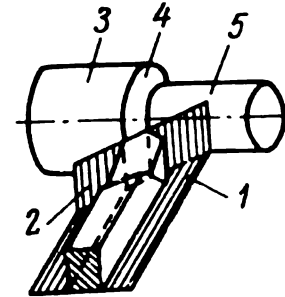


Fig. 184. Principal planes and surfaces in metal cutting:

1—basic plane; 2—cutting plane; 3—work surface; 4—cutting surface; 5—machined surface

allel to the longitudinal and cross feeds. In a lathe tool the basic plane coincides with the base of the tool.

Single-point tool angles are classified as chief, auxiliary and plan angles.

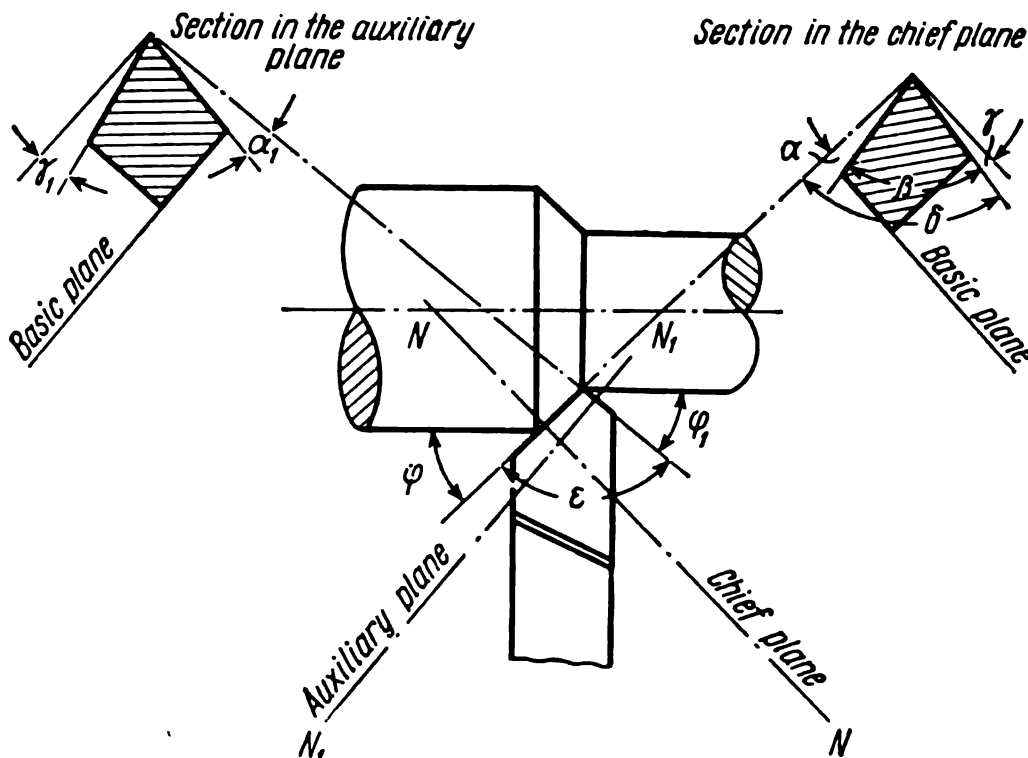


Fig. 185. Single-point tool angles

In the chief plane  $N-N$  (Fig. 185), which is perpendicular to the projection of the side-cutting edge on the basic plane, we may distinguish the side-relief angle  $\alpha$ , side-rake angle  $\gamma$ , lip angle  $\beta$  and the cutting angle  $\delta$ .

The *side-relief angle*  $\alpha$ , between the side flank and cutting plane, serves to reduce the friction between the cutting surface and the side flank; it may range from  $6^\circ$  to  $12^\circ$ .

The *side-rake angle*  $\gamma$  is the angle between the tool face and a plane perpendicular to the cutting plane and passing through the side-cutting edge of the tool. The side-rake angle facilitates chip forming; it may be either positive or negative and ranges from  $-10^\circ$  to  $+15^\circ$ . The softer the material being machined the larger angle  $\gamma$  should be.

The angles measured in the auxiliary plane  $N_1-N_1$  which is perpendicular to the projection of the end-cutting edge on the basic plane are end-relief angle  $\alpha_1$  and the back-rake angle  $\gamma_1$ . The *end-relief angle*  $\alpha_1$  is between the end flank and a plane passing through the end-cutting edge and perpendicular to the basic plane. It is taken from  $8^\circ$  to  $10^\circ$  thereby ensuring minimum friction between the tool and the machined surface.

The plan angles are the plan approach, or entering, angle  $\varphi$ , the end-cutting-edge angle  $\varphi_1$  and the nose angle  $\epsilon$ . These tool angles affect the tool life and the permissible cutting speed.

The *plan approach angle*  $\varphi$  is the angle between the projection of the side-cutting edge on the basic plane and the direction of feed; it may vary from  $30^\circ$  to  $70^\circ$  and it usually equals  $45^\circ$  for metals of medium hardness.

The *end-cutting-edge angle*  $\varphi_1$  is the angle between the projection of the end-cutting edge on the basic plane and the direction of feed. As a rule it is taken from  $10^\circ$  to  $15^\circ$ .

The *nose angle*  $\epsilon$  is formed by the projections of the side- and end-cutting edges. The larger angle  $\epsilon$  is, the longer the tool life will be. Angle  $\epsilon$  is found from the equation  $\epsilon = 180^\circ - (\varphi + \varphi_1)$ .

### 31-3. Chip Formation and Types of Chips

**The chip formation process.** If the work  $A$  is clamped in a machine tool, and the single-point tool  $B$  is set to a certain depth of cut  $t$  and moved in the direction of the arrow (Fig. 186a) under the action of force  $P$  then, after coming into contact with the work, the face of the tool will gradually be forced into the metal and will compress its surface layer. The pressure of the tool will cause elastic strain in the surface layer, i. e., the latter will be in a stressed state. If we continue to force the tool into the work, a condition will be reached in which the stress first exceeds the elastic limit and then the ultimate strength of the given metal. This leads to shear along the shear plane  $N-N$  and the cutting off of an element of the chip from the work (Fig. 186b).

Under the action of the same forces and in the same way the second, third and subsequent chip elements are separated from the work and

form the chip (Fig. 186c). The second and following chip elements are sheared off along shear planes parallel to plane  $N-N$ .

The angle between the shear plane and the machined surface is the shear angle  $\Delta$ . It has been established that the shear angle for various metals varies in a range from  $145^\circ$  to  $155^\circ$ .

The chip formed in the metal cutting process undergoes plastic deformation—it becomes shorter and its cross section increases, this is called chip contraction. Due to contraction the length of the chip obtained will be much shorter than the length of travel of the tool along the surface of the work.

**Types of chips.** The three types of chips obtained in cutting various metals are continuous, sheared and discontinuous, or segmental, chips (Fig. 187).

A *continuous chip* (Fig. 187a) is formed in machining soft ductile metals (low-carbon steel, copper, aluminium, lead, etc.) with small feeds, high cutting speeds and a large rake. The machined surface has a smooth glossy appearance.

A *sheared chip* (Fig. 187b) is formed in machining less ductile metals (high-carbon steels) with heavy feeds (large chip thickness), low cutting speeds and a small rake. An even machined surface will be obtained.

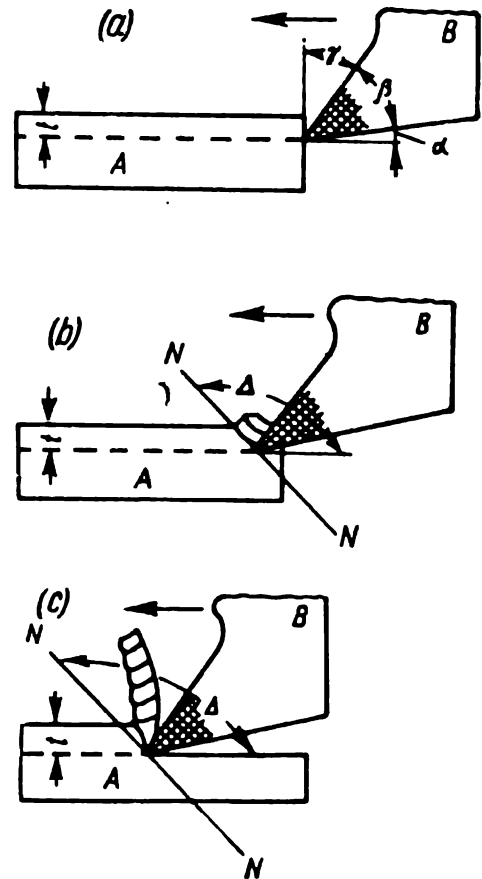


Fig. 186. Process of chip formation

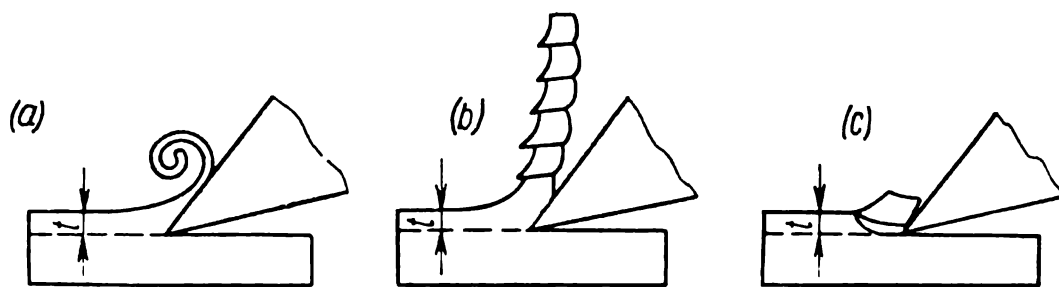


Fig. 187. Types of chips

*Discontinuous, or segmental, chips* (Fig. 187c) consist of separate particles of metal of irregular shape and are formed in machining brittle metals, such as cast iron and bronze. More or less evident traces of roughness are found on the machined surfaces.



### 31-4. Forces Acting on a Single-point Tool

In machining, the metal being cut offers resistance to the cutting tool. This resistance is overcome by the cutting force applied to the tool face. The work done by this force in cutting is expended in shearing the chip from the work, deforming the chip, and overcoming the friction of the chip on the tool face and the tool flank on the cutting surface.

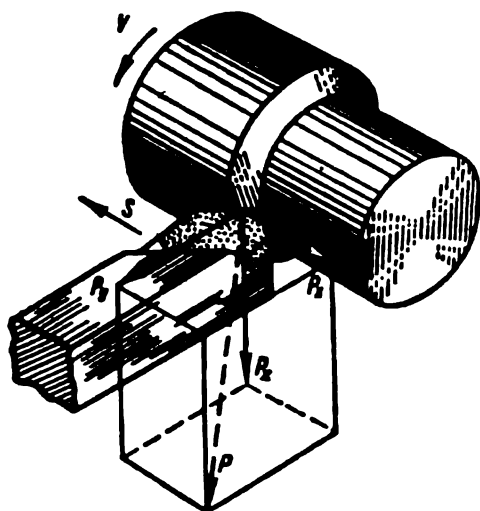


Fig. 188. Component forces acting on a turning tool

The magnitude of the cutting force depends upon the properties of the metal or alloy being machined, the rate of feed and depth of cut, tool angles, cutting speed, coolant used and a number of other factors.

The cutting force  $P$  in straight turning is usually resolved into three components:  $P_x$ ,  $P_y$ , and  $P_z$  (Fig. 188).

Component force  $P_x$  acts in a horizontal plane parallel to the work axis; it is called the axial, or *feed, force*.

The component force  $P_y$  acts in a horizontal plane along a radius of the work and is called the *radial force*.

The component force  $P_z$  acts in a vertical plane tangent to the cutting surface in the direction of the primary cutting motion. It is called the vertical tangential force or simply the *cutting force*.

The resultant of the three component forces is determined by the following formula from mechanics:

$$P = \sqrt{P_x^2 + P_y^2 + P_z^2}. \quad (102)$$

Certain experimental data establish the following approximate ratios between the feed force  $P_x$ , radial force  $P_y$ , and the cutting force  $P_z$ :  $\frac{P_x}{P_z} = 0.3$  to  $0.4$  and  $\frac{P_y}{P_z} = 0.4$  to  $0.5$ .

The largest of the three component forces is  $P_z$ . It develops the torque  $M_t$  on the workpiece according to the formula

$$M_t = \frac{P_z D}{2} \text{ kg-mm}, \quad (103)$$

where  $D$  = work diameter, in mm.

The machine tool, cutting tool and workpiece are subject to loads during the metal cutting process due to the action of the three mutually perpendicular forces considered above. This leads to elastic deformation of the tool and the work, and their resulting deflection in the direction of force action is one of the reasons for inaccuracies in

machining. Thus, the cutting force  $P_z$  bends the tool downwards and the tool, under the action of this force, tends to bend the work upwards; the force  $P_x$  bends the tool in a direction opposite to the longitudinal feed and tends to reduce the rate of feed while force  $P_y$  pushes the tool away from the work and tends to reduce the depth of cut.

Taking into consideration the forces  $P_x$ ,  $P_y$  and  $P_z$ , the cross-sectional area of the chip is commonly reduced in finishing passes to obtain higher machining accuracy. Proper selection of the tool angles and the application of cutting fluids also promote an improvement in the quality and accuracy of machining.

### 31-5. Heat Generated in the Metal-cutting Process. Tool Life and Wear

All the mechanical work done in cutting metal is converted into an equivalent amount of heat.

The work  $A$  done in cutting depends upon the cutting force  $P_z$  and the cutting speed  $v$ . It is determined from the formula

$$A = P_z v \text{ kg-m per min.} \quad (104)$$

The amount of heat  $Q$  generated in a unit of time (one minute) by metal cutting is based upon the thermal equivalent of work, equal to 427 kg-m per kcal; then

$$Q = \frac{A}{427} = \frac{P_z v}{427} \text{ kcal per min.} \quad (105)$$

The generated heat is distributed between the workpiece, chip and tool; only a negligible amount of heat is dissipated to the ambient air.

Tool life and tool wear are in a direct relationship with the amount of heat generated in metal cutting and its distribution between the work, chip and tool and, consequently, with the cutting temperature.

Tool life is the time a tool will operate continually at given cutting conditions until it is dulled. The main factor affecting tool life is the cutting speed.

The rate of tool wear depends upon the properties of the tool and workpiece materials, cutting speed and temperature and a great many other factors.

### 31-6. Influence of Various Factors on the Cutting Speed. High-velocity and Heavy-feed Metal Cutting

*The cutting speed* permitted by a tool depends on a number of factors, chief of which are: (1) mechanical properties of the material being machined, (2) properties of the tool point material, (3) tool life, (4) rate of feed, (5) depth of cut, (6) tool angles and (7) type of cooling employed.

Insofar as the cutting conditions are concerned, the most important properties of the material being machined are the tensile strength  $\sigma_b$  and the hardness Bhn. At a constant tool life an increase in the tensile strength and hardness of the work material requires a decrease in cutting speed to avoid premature wear of the tool.

The effect of the cutting speed on the tool life is taken into consideration by a correction factor. A tool life  $T$  equal to 30 minutes is taken as unity for a given cutting speed. Upon an increase in the cutting speed the tool life will be less than unity and upon a decrease it will be more. This relationship is based upon the amount of heat generated in a unit of time; at high cutting speeds more heat is generated, thereby reducing the cutting properties of the tool, and vice versa.

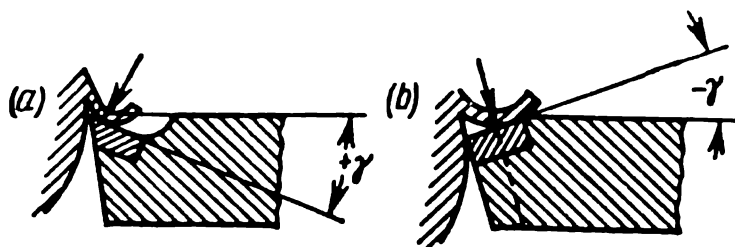


Fig. 189. Cutting force acting on the face of a carbide-tipped tool:

(a) with positive rake; (b) with negative rake

For a given tool life an increase in the feed and/or the depth of cut leads to a reduction in the permissible cutting speed.

The plan approach angle  $\phi$ , side-rake angle  $\gamma$  and the side-relief angle  $\alpha$  greatly affect the cutting speed. Under constant cutting conditions an increase in angle  $\phi$  correspondingly increases the thickness of the undeformed chip and reduces its width. This obviously reduces the length of the side-cutting edge actually engaged in the cutting process, thereby increasing the amount of heat generated per unit of edge length and reducing the tool life. On the contrary, a decrease in angle  $\phi$  increases tool life. Thus, if angle  $\phi$  is reduced from  $90^\circ$  to  $30^\circ$ , the cutting speed can be increased almost twofold for a given tool life.

In machining soft metals an increase in the side-rake angle  $\gamma$  will increase tool life within definite limits, since this reduces chip deformation and the cutting force, and thus enable the cutting speed to be increased. Very hard materials can be expediently machined with carbide-tipped tools having a negative side-rake angle  $\gamma$ . This feature changes the directions and magnitudes of the forces acting on the tool and promotes higher tool life. With a positive rake the tool point is subject to a bending force; with a negative rake—to a compressive force (Fig. 189). Tool life will be higher in the second case.

The optimum value of the side-relief angle, enabling maximum cutting speeds to be applied for a given tool life, varies from  $6^\circ$  to  $15^\circ$  depending upon the workpiece material.

The tool and work may be cooled during machining to lower the cutting temperature. This increases the tool life and, consequently, permits the cutting speed to be increased. Thus, coolant feed at a rate of 8 to 12 litres per minute enables the cutting speed to be increased by 15 to 25 per cent for roughing operations and 5 to 8 per cent for finishing.

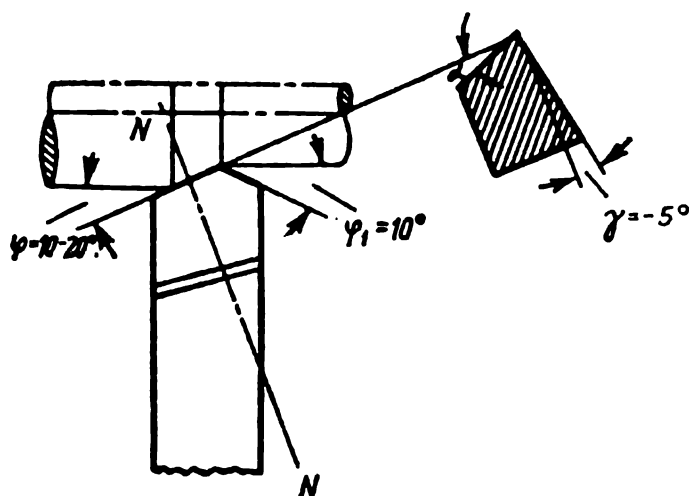


Fig. 190. KBYK single-point tool for high-velocity turning of low-machinability steels

Cutting fluids used in metal cutting include water-base soda and soap solutions, emulsion-type soluble oils, vegetable and mineral oils (linseed, rape, spindle and solar oils, as well as sulphurised oils containing sulphur as an active agent).

At present, heat-resistant alloys and other materials with a low machinability are being cooled with a high-pressure jet. The cutting fluid is applied at a high pressure (20 to 30 atm) through a narrow slit in the coolant hose nozzle along the tool flank from underneath. This method provides more intensive heat disposal and reduces friction on the flank of the tool.

High-pressure jet cooling increases tool life from 5 to 7 times in comparison with conventional cooling methods and, therefore, enables the cutting speed to be considerably increased.

**High-velocity and heavy-feed metal cutting.** On the basis of the effect of tool geometry on permissible cutting speeds, progressive Soviet lathe operators and tooling engineers have devised new cutting tools that enable them to apply high-velocity and heavy-feed metal cutting procedures. The aim of their work in this field is to fully utilise the cutting properties of carbide- and ceramic-tipped tools, as well as the available power of the machine tool and process-

ing facilities. One example is the high-production KBYK\* tool used in turning low-machinability steels. This tool has a small plan approach angle and negative rake (Fig. 190) which support the cemented carbide tip better, improve heat removal from the cutting edges and

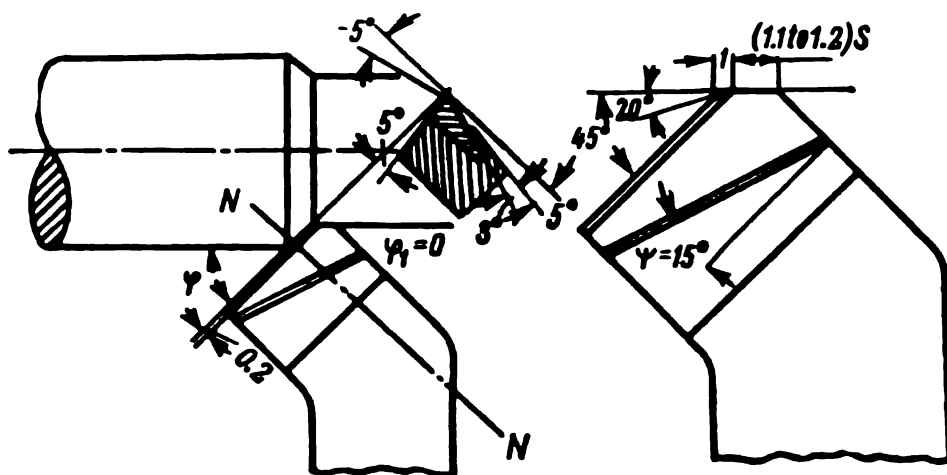


Fig. 191. Single-point tool designed by V. Kolesov

generally increase tool life. All of these features allow the cutting speed in machining, steel 45, for example, to be raised to 1500 m per min.

Higher cutting speeds are only one measure for increasing the rate of production in machining. An analysis of the machining time formula (97) reveals a second method, e.g., by increasing the rate of feed. This opportunity was used by V. Kolesov, a highly skilled lathe operator of the Srednevolzhsky Machine Tool Plant, who employed a single-point tool (Fig. 191) of ingenious design for high-velocity metal cutting with heavy feeds.

The rate of production is increased, when the Kolesov tool is applied, by an increase in the feed from 0.1-0.6 to 1-3 mm per revolution, i.e., 5- to 10-fold. Moreover, the roughing and finishing passes are combined, for which purpose the end-cutting-edge angle is reduced from the usual  $10^\circ$  or  $15^\circ$  to zero.

Fig. 192. Effect of the end-cutting-edge angle on the ridges formed in turning

A zero end-cutting-edge angle proves highly efficient; in a tool of this type the end-cutting edge cleans up the turned surface, removing the ridges and other irregularities left on the surface by tools of conventional design (Fig. 192).

\* Named after its four designers—V. Krivoukhov, B. Brushtein, S. Yegorov, and N. Kozlov.

In this way, the tool life, cutting speed and feed can be increased substantially and the finish of the machined surface improved by varying the tool angles.

### 31-7. Power Required in Metal Cutting

The effective power  $N_c$  required in metal cutting depends upon the cutting speed  $v$  and the cutting force  $P_z$ ; thus

$$N_c = \frac{P_z v}{60 \times 75} \text{ hp}, \quad (106)$$

or

$$N_c = \frac{P_z v}{60 \times 75 \times 1.36} \text{ kW}. \quad (107)$$

The net power that must be supplied to the machine tool must take into account the efficiency of the latter:

$$N_m = \frac{N_c}{\eta_{mt}} = \frac{P_z v}{60 \times 102 \eta_{mt}} \text{ kW}, \quad (108)$$

where  $\eta_{mt}$  = efficiency of the machine tool.

The efficiency indicates the power lost in overcoming friction resistances in the main drive and feed mechanisms.

Experimental methods were employed to determine the following mean efficiency values at full load for various machine tools: 0.8 to 0.9 for lathes, 0.85 to 0.9 for drilling machines, 0.8 to 0.9 for milling machines, 0.65 to 0.75 for shapers and planers and 0.8 to 0.85 for grinding machines.

### 31-8. Machine Tool Classification

From the point of view of their field of application machine tools may be classified as: general-purpose, including the lathe, planer, shaper, and drilling, milling and grinding machine groups; single-purpose—broaching, thread-cutting and other machines; specialised—unit-type, or unit-built, machines; automatic machine tools and others (centring, filing, etc.).

Within each group the machine tools are subdivided into types and models in accordance with their design and processing features, degree of specialisation, etc.

Machine tool models are designated by a number. Soviet machine tool plants use a two-digit code system in which the first figure indicates the group of machine tool and the second—its characteristic feature (type, model, etc.).

## Chapter 32

### LATHE PRACTICE

#### 32-1. The Lathe

About one half of the machine tools operating in engineering plants are of the lathe group.

On the basis of their purpose, design, number of tools accommodated, degree of mechanisation and other factors, lathe-type machine tools may be classified as: (1) engine lathes (single-tool), (2) multiple-tool lathes, (3) turret lathes, (4) vertical turret lathes, and vertical turning and boring mills, (5) semiautomatic and automatic lathes and (6) special-purpose lathes.

The construction and principle of lathe-type machine tools will be illustrated by the example of the most common representative of this class—the engine lathe.

The principal units of an engine lathe (Fig. 193) are: the bed, headstock, tailstock, quick-change gearbox (feed gearbox) with the lead screw and feed rod or bar, apron with the feed mechanism, carriage and electrical drive. Besides these units the lathe may have a pump for cutting fluid delivery and push button or lever controls for starting and stopping operation.

The primary cutting motion and the feed motion are accomplished by means of the corresponding drives which are systems of mechanisms for transmitting power from its source (the electric motor) to an operative unit of the lathe (spindle, carriage, etc.). An engine lathe commonly has two drives—the main drive and the feed drive.

**Main drive.** The main drive (Fig. 194) consists of the motor 1, belt drive 2, speed gearbox 3 and spindle 4. By means of this drive which includes the motor driving pulley, belt and the driven pulley of shaft 1-1, power is transmitted to the first operative unit of a lathe—the spindle (shaft 11-11). The speed gearbox enables the most expedient cutting speeds to be employed by changing the spindle speed  $n$ .

Two general types of speed gearboxes are used on lathes; one provides stepped and the other infinitely variable spindle speeds. This means that with the first type the maximum, minimum and a series of intermediate spindle speeds in definite steps are available. Infinitely variable gearing systems enable any spindle speed from the minimum to the maximum to be obtained. This advantageous feature allows the most suitable cutting speed for each workpiece

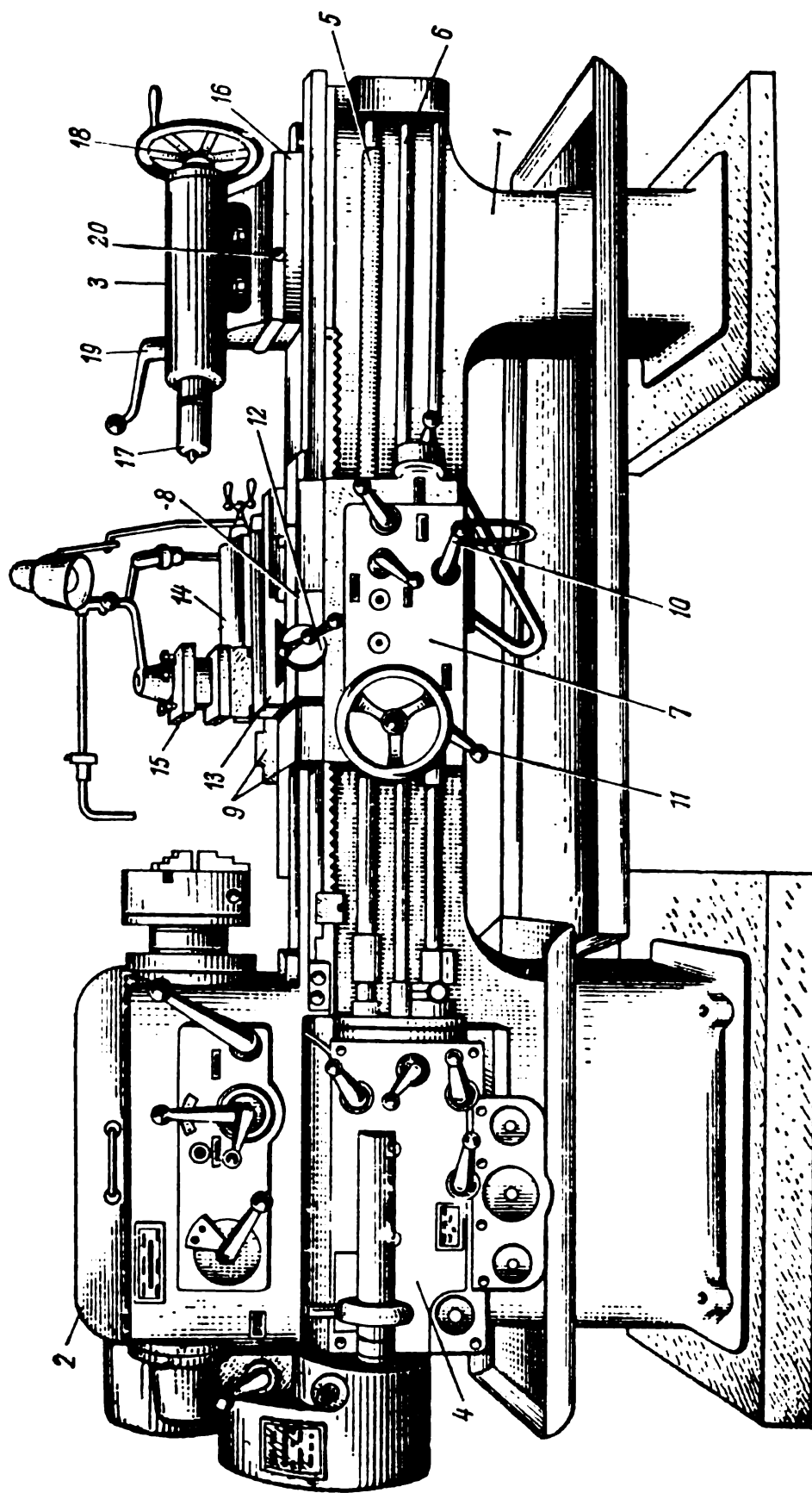


Fig. 193. Engine lathe, model 1A62:

1—bed; 2—headstock; 3—tailstock; 4—quick-change gearbox; 5—lead screw; 6—feed rod; 7—apron; 8—carriage; 9—saddle; 10—lever for engaging longitudinal power feed of the carriage; 11—handwheel; 12—ball-crank handle for manual cross feed of the cross slide; 13—cross slide; 14—compound rest; 15—square turret; 16—tailstock base; 17—tailstock spindle; 18—handwheel; 19—binding lever for the tailstock spindle; 20—set-over screw



diameter to be set up, thereby maintaining the specified surface finish without decreasing the rate of production.

*Stepped spindle speed variation* in an engine lathe may be achieved by either a geared headstock (Fig. 194) or a cone pulley. A geared headstock is more efficient and compact than other types of lathe drives. Power is transmitted from the motor through a belt drive to shaft *I-I* which carries the sliding triple cluster gear with three

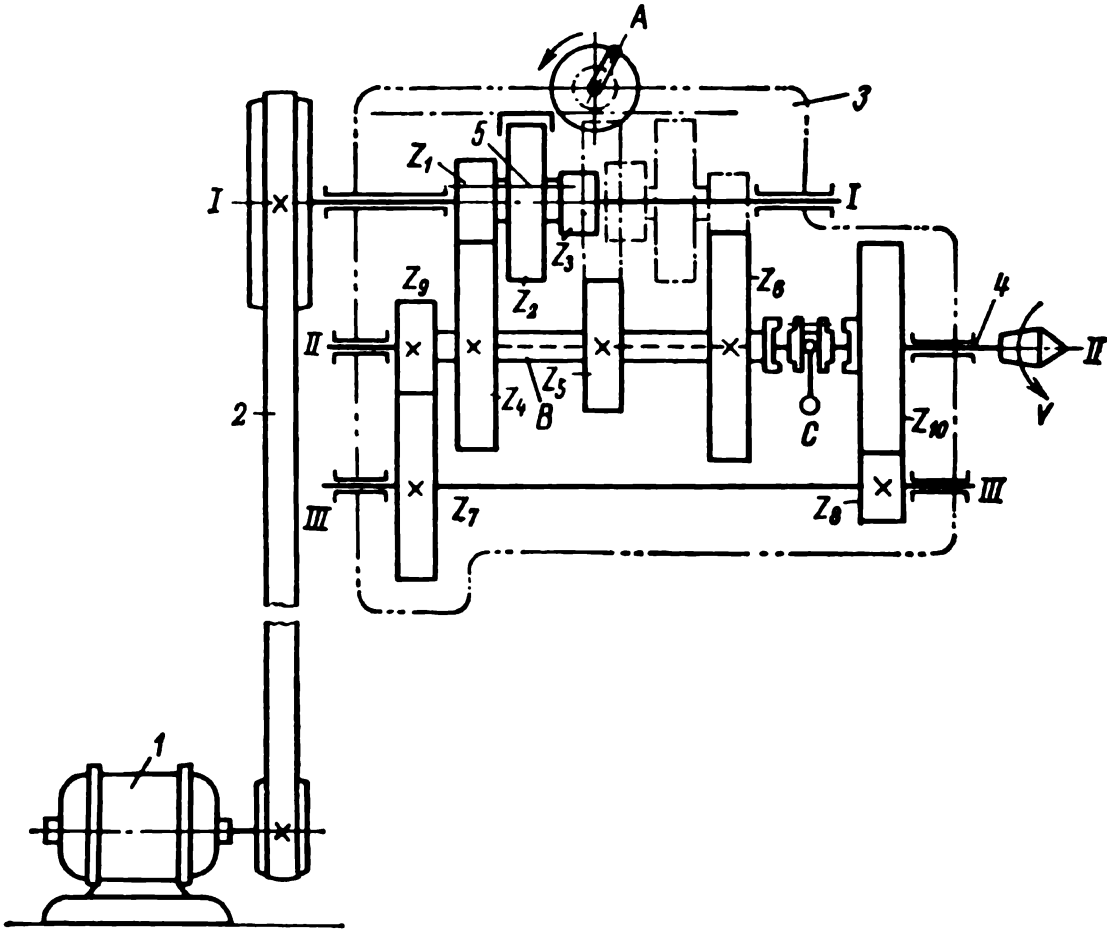


Fig. 194. Main drive of a lathe with a geared headstock

rims  $z_1$ ,  $z_2$  and  $z_3$ . By means of lever *A* the cluster gear can be shifted on key 5 along shaft *I-I* so that its three rims mesh in turn with the gears  $z_4$ ,  $z_5$  and  $z_6$  which are rigidly mounted on sleeve *B*. This sleeve is freely mounted on the lathe spindle *II-II*. The gears  $z_9$  and  $z_{10}$ , mounted directly on the spindle, are constantly in mesh with gears  $z_7$  and  $z_8$  of the countershaft *III-III*. When clutch *C* is shifted into engagement to the left, countershaft *III-III* is disengaged and the spindle obtains one of three speeds— $n_1$ ,  $n_2$  and  $n_3$ —depending upon the position of the sliding cluster gear.

If the speed of shaft *I-I* is denoted by  $n_0$  then

$$n_1 = n_0 \frac{z_1}{z_4}, \quad n_2 = n_0 \frac{z_2}{z_5}, \quad \text{and} \quad n_3 = n_0 \frac{z_3}{z_6}, \tag{109}$$

where  $\frac{z_1}{z_4}$ , etc. = gearing ratios, indicating the ratio of the number of teeth on gears on the driving shaft (in our case, shaft I-I) to the number of teeth on gears on the driven shaft (spindle II-II).

When clutch C is shifted into engagement to the right, countershaft III-III with a gearing ratio of  $i_{cs} = \frac{z_{10}}{z_8} \times \frac{z_7}{z_9}$  is included in the gear train so that the spindle obtains three more speeds. Therefore, the countershaft enables additional speeds to be obtained.

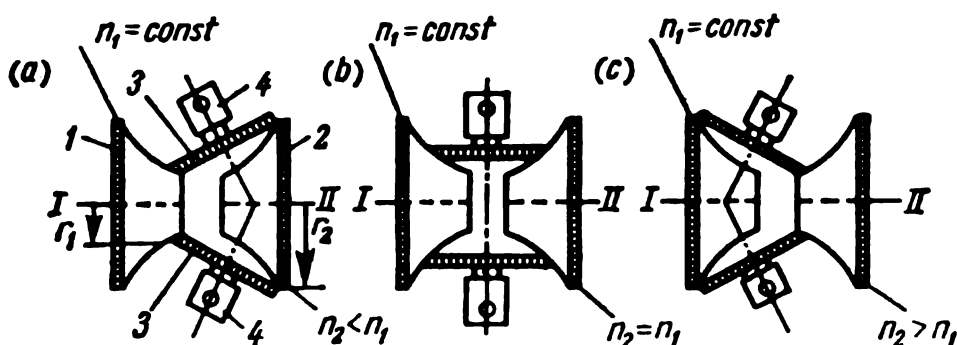


Fig. 195. The Svetozarov friction drive

The simple geared headstock with a countershaft considered here provides six different spindle speeds. Geared headstocks are commonly designed for 3, 4, 6, 8, 12, 16 and 24 spindle speeds.

In properly designed speed-changing systems the values of the spindle speeds in the available range vary in a geometrical progression in which the constant ratio or multiplier  $\varphi$  has one of the following values: 1.06, 1.12, 1.26, 1.41, 1.58 and 2. The less the constant ratio  $\varphi$ , the less the difference will be between successive speeds in the progression and the more exactly the required speed can be approximated.

The advantage of a geared headstock in the main drive is the possibility of transmitting high power; the available power on the spindle remains almost constant for the different speeds.

Lathe headstocks with *infinitely variable* spindle speeds employ either conical pulleys or friction, electrical, electromechanical or hydraulic drives.

In the new powerful Soviet engine lathe, model 1620, the spindle speeds are changed by means of a friction drive invented by V. Svetozarov. This drive provides infinitely variable spindle speeds in a range from 18 to 3000 rpm and longitudinal feeds ranging from 0.05 to 2 mm per revolution.

The schematic diagrams in Fig. 195 illustrate three characteristic positions of the friction cones and rollers in the Svetozarov drive—at minimum, medium and maximum spindle speeds.

The steel friction cones *1* and *2* with spherical surfaces are mounted on hollow shafts *I* and *II*. Shaft *I* is linked to the shaft of a flange-mounted motor through a ball-type clutch. Thus, cone *1* is driven by the motor at a constant speed ( $n_1 = \text{const}$ ). Compressed springs arranged inside of shafts *I* and *II* force cones *1* and *2* against the steel rollers *3-3*. Each of the rollers *3-3* can be swivelled together with shaft *4* through a certain angle.

When the motor is switched on, cone *1* rotates with its shaft and, since it is held firmly against the rollers, it rotates them by friction. The rollers transmit this rotation, also by friction, to cone *2*. The

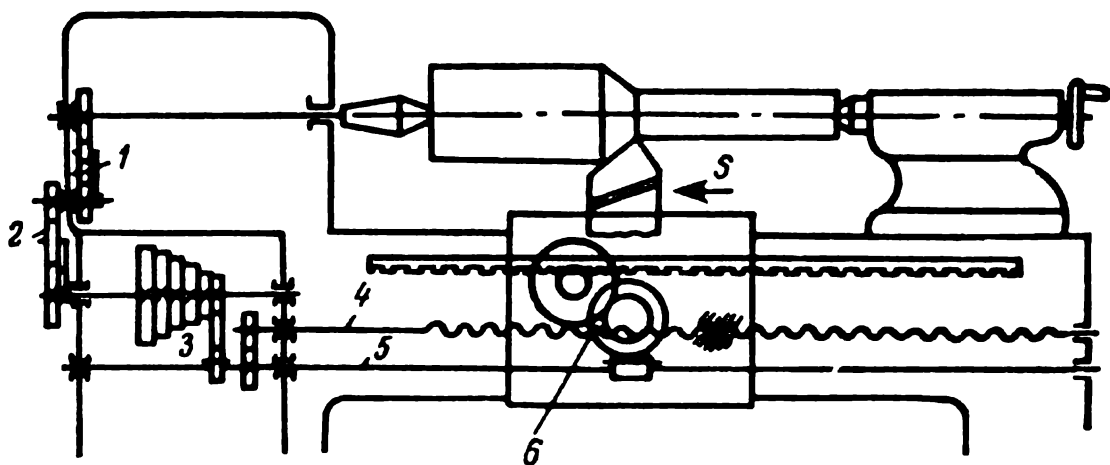


Fig. 196. Schematic diagram of an engine lathe feed drive

speed  $n_2$  of cone *2* depends upon the ratio of the active radii  $r_1$  and  $r_2$  and is determined from the formula

$$n_2 = n_1 \frac{r_1}{r_2}. \quad (110)$$

For the position of the rollers shown in the first case (Fig. 195a) the ratio of the active radii  $\frac{r_1}{r_2} < 1$ , therefore,  $n_2 < n_1$ . In the second case (Fig. 195b) the ratio is equal to unity and  $n_2 = n_1$ , while in the last case (Fig. 195c) it is more than unity and  $n_2 > n_1$ .

The speed  $n_2$  at which shaft *II* rotates is transmitted to a vee-belt pulley and further to the spindle. By changing the position of the rollers along the spherical surface of cones *1* and *2*, spindle speeds can be infinitely varied from the minimum to the maximum values.

The Svetozarov drive can provide spindle speed adjustment in the range  $\Delta = \frac{n_{\max}}{n_{\min}} = 4$  to 8. If this friction drive is employed in conjunction with a three-stage geared headstock a still wider range of infinitely variable speeds can be obtained.

**Feed drive.** The feed drive serves to transmit power from the spindle to the second operative unit of the lathe—the carriage, i.e., it

converts rotary motion into linear motion of the tool and also enables the specified rate of feed to be selected as well as its direction. The feed drive not only provides the different feeds required in turning but also provides a wide range of pitches for thread-cutting operations.

The feed drive is usually obtained from the last link of the main drive gear train—the spindle. In this case the gear of the reversing

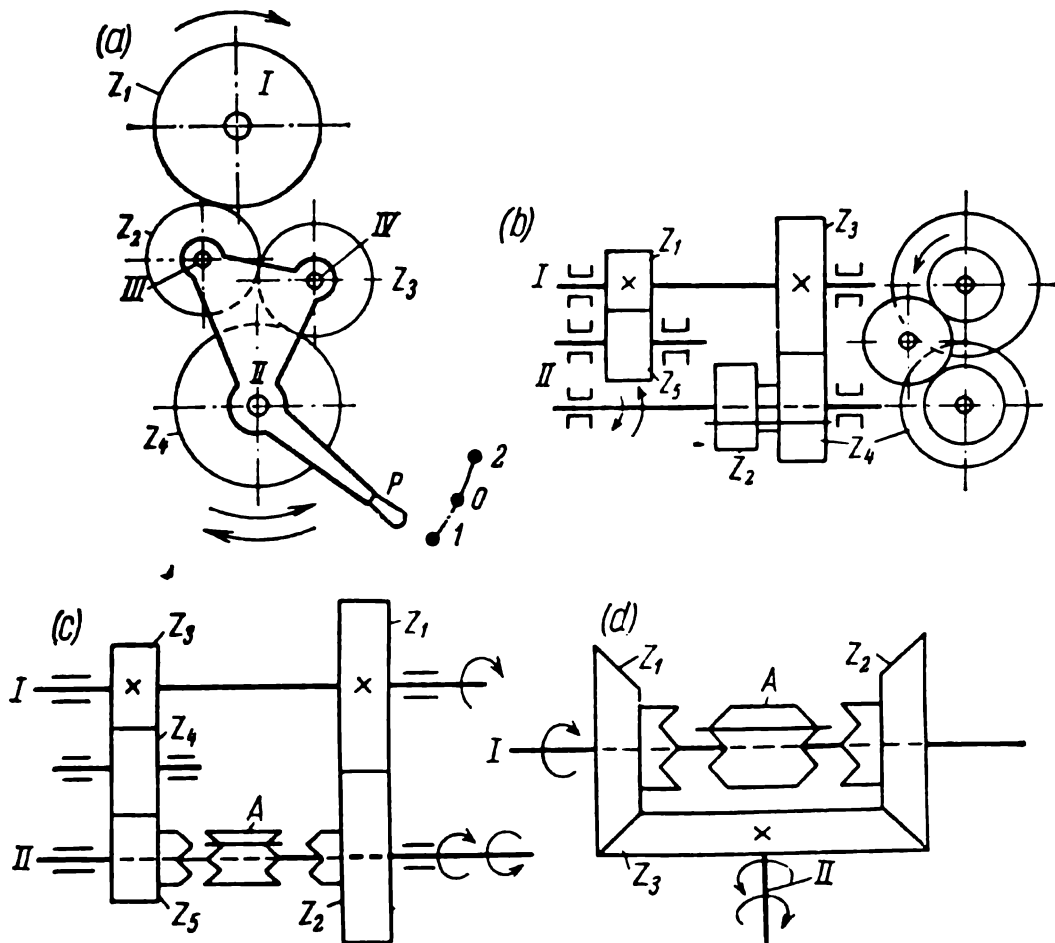


Fig. 197. Reversing mechanisms of engine lathes

mechanism is mounted either on the spindle or on an intermediate shaft linked to the spindle by separate gearing. Sometimes the feed drive is powered from a separate motor.

The feed drive of an engine lathe (Fig. 196) consists of the reversing mechanism 1, change-gear quadrant 2, quick-change gearbox 3 with lead screw 4 and feed rod 5, and apron 6.

The purpose of the *reversing mechanism* is to change the direction of lead screw or feed rod rotation so that the carriage with the cutting tool travels either to the right or to the left, as required.

A reversing mechanism may comprise either

(1) four spur gears with consecutive engagement of the bracket holding the reverse gears (Fig. 197a),

(2) a system of spur gears with parallel engagement and a cluster gear sliding along a key (Fig. 197b),

(3) a system of spur gears with parallel engagement and a jaw clutch (Fig. 197c), or

(4) a system of bevel gears and a jaw clutch (Fig. 197d).

Other arrangements are also employed.

The first of the above-mentioned types is most frequently applied for engine lathes. Here, gear  $z_1$  is mounted on spindle *I* and gear  $z_4$ —on shaft *II* which is usually a shaft of the change-gear quadrant. Gears  $z_2$  and  $z_3$  are freely mounted on studs *III* and *IV* of the bracket which also has the lever *P*. The bracket is mounted on shaft *II* and can be turned by the lever to position 1, 0 or 2. Gears  $z_2$ ,  $z_3$  and  $z_4$  are constantly in mesh with each other.

If lever *P* is shifted to position 1, gears  $z_1$ ,  $z_2$ ,  $z_3$  and  $z_4$  are put into consecutive mesh and shaft *II* begins to rotate in a direction opposite to that of the spindle. In position 2 of lever *P* only gears  $z_1$ ,  $z_3$  and  $z_4$  will be in mesh and shaft *II* will rotate in the same direction as the spindle. In position 0 both gears  $z_2$  and  $z_3$  are pulled out of mesh with gear  $z_1$  and no rotation is transmitted from the spindle to shaft *II*; the reversing bracket is disengaged.

When gears  $z_3$  and  $z_4$  are in mesh in a reversing mechanism with a sliding cluster gear (Fig. 197b) shaft *II* is rotated in the direction opposite to that of shaft *I*; if the cluster gear is shifted so that gears  $z_1$ ,  $z_3$  and  $z_2$  are in mesh, then the direction of rotation of shafts *I* and *II* will coincide.

Reversing is done in a similar manner in mechanisms with a jaw clutch (Fig. 197c and d).

The *change-gear quadrant* of an engine lathe consists of a set of change gears and the device known as the quadrant proper. The change-gear quadrant serves to set up the feed drive to different speeds of the lead screw and feed rod which, in turn, provide for different rates of feed in turning and different pitches in cutting threads. The quadrant also transmits motion to the quick-change gearbox or to the lead screw.

Change gears used for engine lathes may be either single-pair, two-pair or three-pair arrangements.

In the single-pair gearing shown in Fig. 198a gear  $z_1$  is mounted on shaft *I* of the reversing mechanism and gear  $z_3$ —on shaft *II* of the quick-change gearbox. The intermediate gear  $z_2$  is mounted on stud *E* which can be adjusted and clamped as required along a straight slot in quadrant *D*. The latter is mounted on shaft *II* and can turn freely in reference to this shaft. The gearing ratio of the quadrant is determined by the ratio of the numbers of teeth on gears  $z_1$  and  $z_3$ . After changing gears  $z_1$  and  $z_3$  for a new job, the intermediate gear is made to mesh with these two gears by adjusting stud

$E$  along the straight slot and turning quadrant  $D$  along the circular slot, thereby changing the centre distances between shaft  $I$  and stud  $E$  and between stud  $E$  and shaft  $II$ .

Two-pair gearing (Fig. 198b) has four gears  $z_1$ ,  $z_2$ ,  $z_3$  and  $z_4$  by means of which rotation is transmitted from shaft  $I$  of the reversing device to shaft  $II$  of the gearbox. Gear  $z_1$  is keyed on shaft  $I$ , gear  $z_4$  on shaft  $II$  and gears  $z_2$  and  $z_3$  are keyed on a sleeve which, in turn, is mounted freely on stud  $E$ .

For new gearing ratios with new sets of change gears the centre distances are changed as required by adjusting stud  $E$  along the straight slot and swivelling the quadrant about shaft  $II$  along the circular slot. The gearing ratio of the change gear quadrant is  $i_q = \frac{z_1}{z_2} \times \frac{z_3}{z_4}$ .

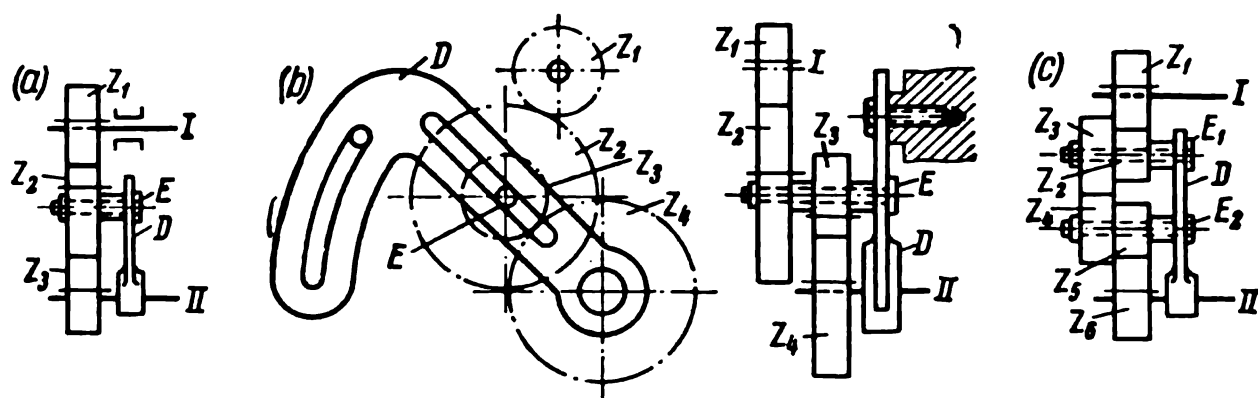


Fig. 198. Change-gear quadrant arrangements

Three-pair gearing (Fig. 198c) consists of six gears  $z_1$  through  $z_6$ . Gear  $z_1$  is secured on shaft  $I$ , gear  $z_6$  on shaft  $II$  while gears  $z_2$  and  $z_3$ , and gears  $z_4$  and  $z_5$  are keyed on two sleeves mounted on studs  $E_1$  and  $E_2$ , respectively. Three-pair gearing offers even more opportunities for setting up an accurate gearing ratio by proper selection of the change gears. The gearing ratio here is  $i_q = \frac{z_1}{z_2} \times \frac{z_3}{z_4} \times \frac{z_5}{z_6}$ .

Standard minimum sets of change gears with standard series of numbers of teeth have been established by computations for engine lathes to ensure normal operation on the most diverse jobs.

*The quick-change gearbox*, located on the front of the lathe under the headstock, allows a variety of feeds to be easily and rapidly selected by merely shifting the corresponding levers, and it is used both for plain turning operations and for thread cutting.

The most widely employed designs of quick-change gearbox are the tumbler-gear and sliding-key types.

A tumbler-gear mechanism (Fig. 199a) comprises a cone of gears  $z_1$  through  $z_6$  mounted on shaft  $II$ , gear  $z_7$  sliding on a key along shaft  $I$ , and arm  $A$ . The tumbler gear  $z_6$  is in constant mesh with gear  $z_7$  and is mounted in arm  $A$ . The drive is usually from the driving shaft

*I* to the driven shaft *II*. The tumbler gear can be brought into mesh with any one of the six gears in the cone by sliding the tumbler arm along shaft *I* and swinging it up or down, according to whether the gear being engaged is small or large. Provision is made for locking the arm in its different positions.

A sliding-key mechanism (Fig. 199*b*) consists of two intermeshing gear cones,  $z_1$  through  $z_4$  and  $z_5$  through  $z_8$ . The first cone is keyed

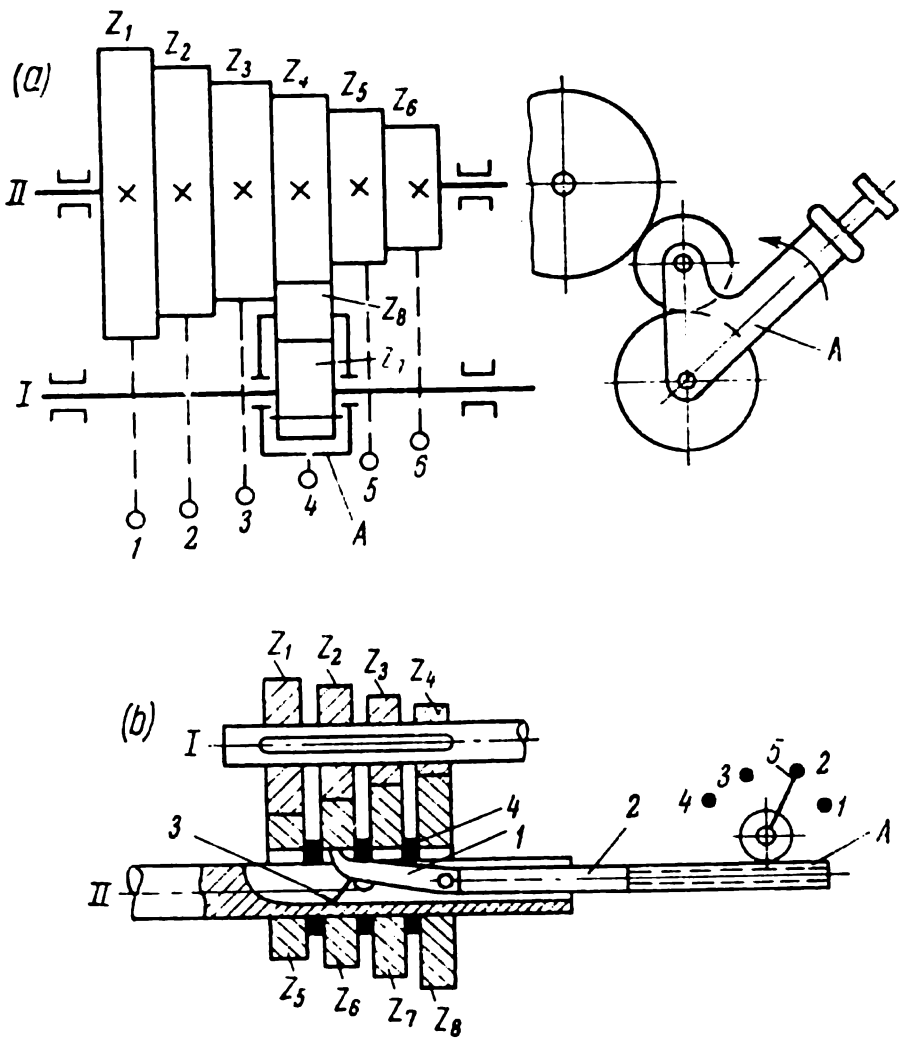


Fig. 199. Quick-change gearbox designs:  
(a) tumbler-gear type; (b) sliding-key type

rigidly on shaft *I* while the gears of the second cone are free to revolve about the hollow shaft *II* except when engaged by the sliding key *I* which can be shifted from one gear to another along the slot of the hollow shaft by means of the tie rod 2 and the rack and pinion mechanism *A*. If lever 5 of this mechanism is turned, the key *I*, under the action of spring 3, enters the keyway of the required gear in the cone  $z_5$ - $z_8$ , thereby providing transmission of rotation from shaft *I* to shaft *II* at the required gearing ratio.

Steel washers or guards 4 without keyways are placed between the gears freely mounted on shaft 11 to prevent simultaneous engagement of two adjacent gears with the key and consequent jamming of the device. Sliding-key mechanisms are reliable in operation and allow feeds to be changed with the gear train running.

*The apron* is secured underneath the saddle of the carriage unit. Its feed mechanisms convert rotary motion of the lead screw or feed rod into linear motion (feed) of the carriage on which the cutting tool is clamped. The apron encloses systems of spur and worm gearing (Fig. 200).

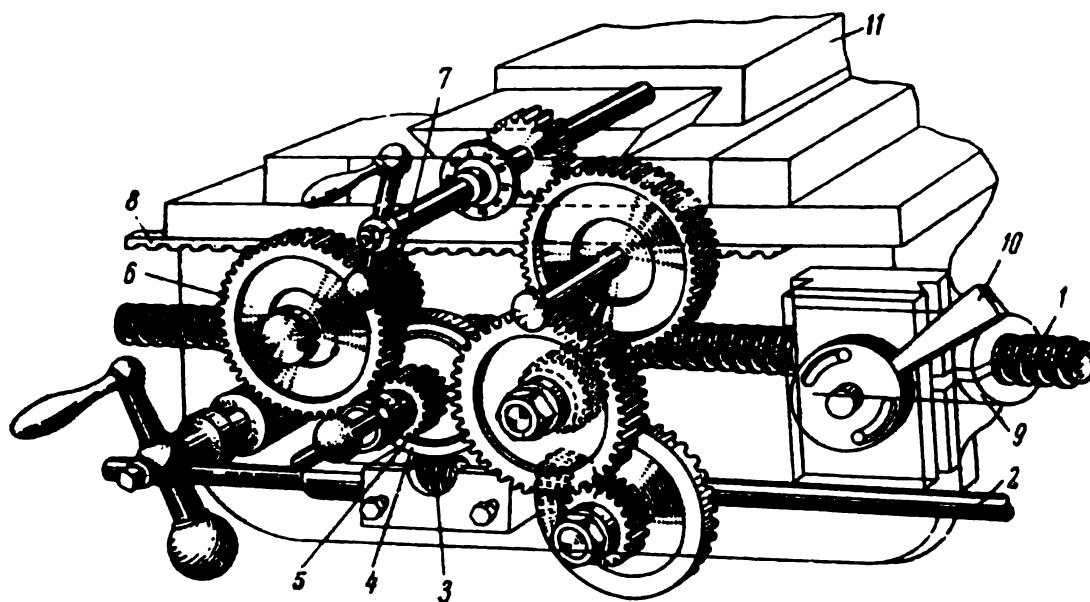


Fig. 200. Lathe apron mechanism

Power is supplied to the apron gearbox by the lead screw 1 only when screw threads are to be cut. For other lathe operations the feed rod 2 is used to feed the carriage. Worm 3 with its key slide along the feed rod which has a keyway along its whole length. Worm 3 is rotated by the feed rod and travels along it with the apron due to the two lugs secured on the rear wall of the apron. Rotation of the worm is transmitted to worm wheel 4 and to spur gear 5 mounted on the same shaft. Gear 5 meshes with gear 6 which transmits rotation to pinion 7 since they are keyed to the same shaft. Pinion 7 runs along the stationary rack 8, fastened to the lathe bed, thus feeding the carriage 11, rigidly secured to the apron, along the bed. In this manner, rotary motion of the feed rod is converted into linear motion of the carriage. Other gearing arrangements in the apron convert the rotary motion into cross feed of the cutting tool by rotation of the cross-feed screw in the cross slide.



In cutting screw threads the lead screw *1* is engaged to the feed mechanism and rotates at a definite speed. Feed rod *2* is disengaged from the apron gearing. The lead screw feeds the carriage lengthways through two half nuts *9* mounted on the rear wall of the apron. Upon engaging lever *10* the half nuts are clamped together so that they engage the rotating lead screw and then the half nuts and, with them, the carriage are fed along the bed. Disengagement of lever *10* spreads the half nuts to release the lead screw and the carriage stops.

The worm and spur gearing of the apron are constant; their component parts are not interchangeable. Therefore, different feeds are obtained only by changing the speed of the lead screw or feed rod, as the case may be, this being accomplished by means of the quick-change gearbox or the change-gear quadrant.

**Carriage.** The purpose of the carriage is to hold the cutting tool and to impart to it either longitudinal or cross feed.

The base of the carriage is the saddle which slides along the ways of the lathe bed (Fig. 193).

Longitudinal power feed of the carriage is engaged by shifting lever *10*; the carriage is fed manually by turning handwheel *11* arranged on the apron. At this, the tool clamped on the carriage travels in a direction parallel to the spindle axis.

The ball-crank handle *12* is rotated for manual cross feed. In this case the cross slide *13* travels in a direction perpendicular to the spindle axis along the dovetail ways of the saddle *9*. Mounted on the cross slide *13* is the swivel base with the compound rest *14* and the tool post *15* (the type of tool-holding device shown in Fig. 193 is called a square turret). Whenever required the compound slide can be swung to any angle by means of the swivel base.

**Tailstock.** The chief function of the tailstock (see Fig. 193) is to support long workpieces ( $L > 4D$ ) between centres; it can also serve to hold such cutting tools as drills, core-drills and reamers. The tailstock consists of the cast-iron body mounted on the base *16*. The latter, together with the body, can be adjusted along the bed ways to suit the length of the work by sliding it to the desired position where it is clamped. On heavy lathes an attachment may be provided for traversing the tailstock.

The body has a bore for the hollow tailstock spindle into which the dead centre is fitted. The tailstock spindle with the inserted centre can be advanced and retracted by means of a screw which is rotated by handwheel *18*. The tailstock spindle is clamped in the required position in the tailstock by means of lever *19*.

The body can be offset in respect to the base in a crosswise direction for taper turning by adjusting the set-over screw *20*.

A running, or ball-bearing, centre is employed in the tailstock for high-velocity turning operations.

### 32-2. Gearing Diagram of an Engine Lathe

Gearing or kinematic diagrams are conventional graphical representations of all the kinematic (gear) trains and their coordination in a machine tool.

An engine lathe has two gear trains: from the motor to the spindle and from the spindle to the carriage. The latter train is customarily depicted in two versions, one for longitudinal feed of the tool (in cutting screw threads and turning) and the other for cross feed.

Each gear train of an engine lathe consists of a series of kinematic pairs of toothed gears or of pulleys. The sequences of the linkages forming a train are usually denoted by figures indicating the number of teeth of the successive gears or the diameters of the pulleys. The sequence of linkages is set down in the following manner: a hyphen is written between elements of a kinematic pair mounted on the same shaft and a multiplication sign between those mounted on different shafts.

Fig. 201 illustrates the gearing diagram of the model 1A62 engine lathe described above. We can follow out the gear train from the motor to the spindle on this diagram. This train links the motor shaft (pulley diameter  $d_1=130$  mm) with the pulley of shaft I (pulley diameter  $d_2=260$  mm) of the lathe headstock which contains shafts II, III, IV, V, VI and VII, and a corresponding set of meshing gears. The structural formula of the given gear train can be concisely expressed as follows:

$$n_{sp} = n_m \frac{130}{260} i_{sg} \eta \text{ rpm}, \quad (111)$$

where  $n_{sp}$  = spindle speed, in rpm,

$n_m$  = motor speed, in rpm,

$i_{sg}$  = variable gearing ratio of the speed gearbox (headstock),

$\eta$  = efficiency of the belt drive (taken equal to 0.98).

The model 1A62 engine lathe has a headstock providing 21 different gearing ratios (for a range of spindle speeds from 12 to 1200 rpm).

The gear trains of the carriage for longitudinal and cross feeds, as well as for thread cutting, can be established in the same manner.

The structural or set-up formula required in setting up a lathe to perform a definite job can be determined by analysing the gearing diagram of the lathe as a whole and its gear trains. The generalised data obtained in this manner are written into the tooling chart which is then hung up on a metal board located near the operator's position.



### 32-3. Single-point Lathe Tools

Single-point tools employed in lathe operations may be classified as straight-turning, facing, cut-off, threading and form tools.

*Straight-turning tools* may be either of the roughing or finishing type.

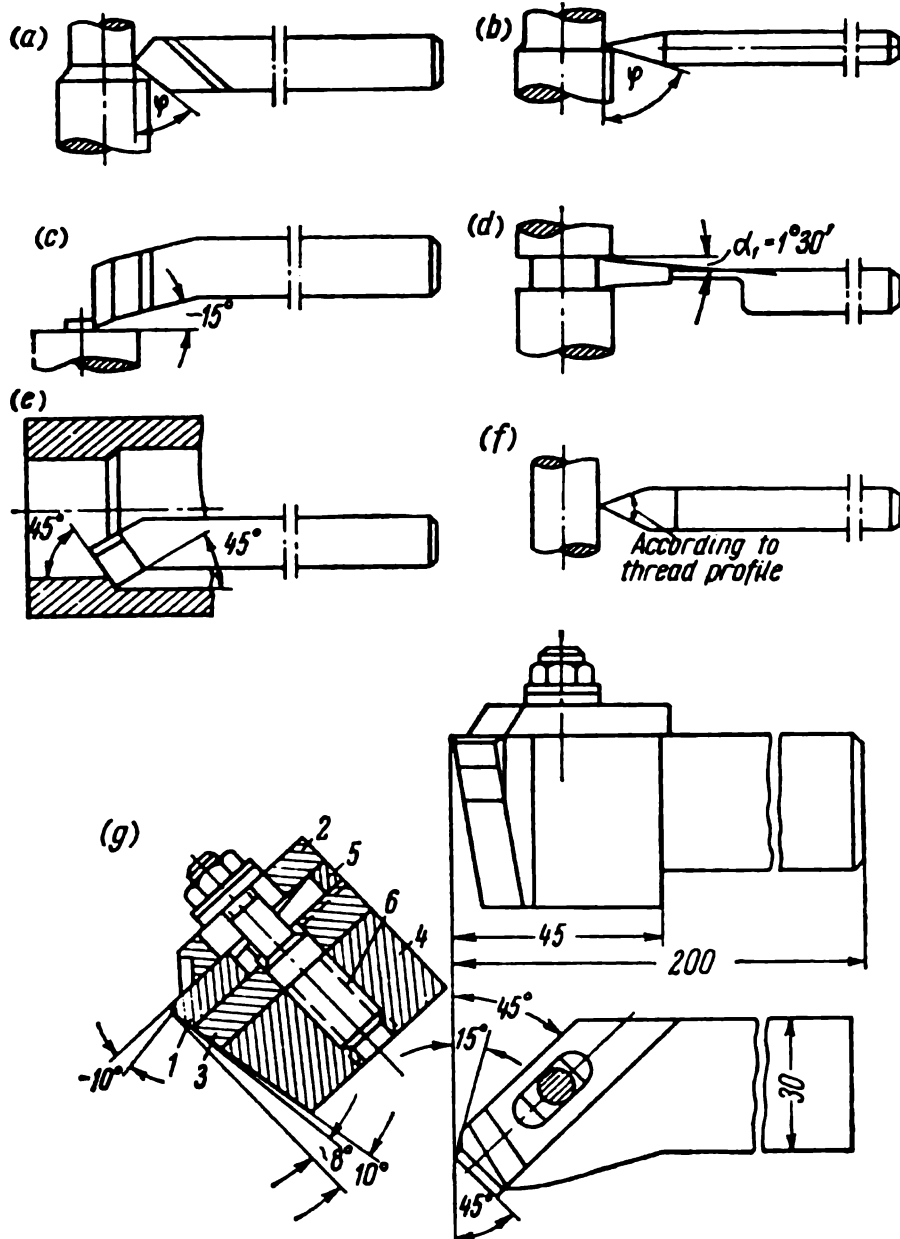


Fig. 202. Single-point lathe tools:

(a) roughing straight-turning straight-shank tool; (b) finishing straight-turning tool with a nose radius; (c) facing tool; (d) cut-off tool; (e) boring tool; (f) threading tool for external threads; (g) lathe tool with mechanically clamped ceramic tip

Roughing straight-turning tools (Fig. 202a), as the name implies, are used for rough turning operations when a heavy chip is to be removed. These tools may be right-hand, left-hand, straight-shank, bent-shank or offset; their cutting angles are selected to attain maximum tool life.

Finishing straight-turning tools (Fig. 202*b*) have a considerably larger nose radius than roughing tools and are used for finishing operations. Broad-nosed finishing tools are often resorted to when an especially fine surface is specified.

Face surfaces on workpieces and shoulders, either square to or at an angle to the workpiece axis, are turned with *facing tools* using a cross feed (Fig. 202*c*).

*Cut-off tools* (Fig. 202*d*) serve to sever the workpiece into parts and also to cut annular grooves.

*Boring tools* (Fig. 202*e*) are employed in machining through and blind holes; their length must always exceed that of the hole being bored.

*Threading tools* (Fig. 202*f*) are of the straight-shank type for external threads and of the bent-shank type for internal threads. The plan angles of these tools must coincide with the thread form.

In addition to those mentioned above special single-point lathe tools are also used in which the tool angles and geometry have been altered to some extent to increase the tool life in turning operations.

It is of interest to note that tools of carbon and alloy steels (and sometimes of high-speed steel) are usually made of a single piece of steel (they are called forged tools) while cemented carbide or ceramic (cemented oxide) tools are of tipped design. Cemented carbide tips are brazed to the shank and ceramic tips are mechanically clamped by various devices. Fig. 202*g* illustrates a clamping arrangement for holding a ceramic tip on a straight-turning tool. Tip 1 is clamped by chip breaker 2 through pad 3 to the shank 4. The position of the tip is reliably fixed by stop 5. Stud 6 with a washer and nut secure the chip breaker to the shank.

### 32-4. Principal Types of Lathe Operations

Lathe-type machine tools, and in particular engine lathes, perform a wide variety of machining operations including turning between centres, in a chuck or on a faceplate; boring; facing; cutting off and cutting grooves (necking); cutting screw threads; taper turning; form turning, and other operations employing appropriate cutting tools and attachments.

**Turning between centres.** Long workpieces (shafts and axles) with a length-to-diameter ratio of  $\frac{L}{D} > 4$  are turned lengthwise between centres, as a rule, using straight-turning tools (Fig. 203). The workpiece, in whose ends centre holes have been previously drilled and on which a driving dog *a* has been clamped, is mounted between the headstock (live) and tailstock (dead) centres. The live centre is inserted into the lathe spindle and the dead centre into the tailstock

spindle. The dog is clamped on one end of the work with screw 1 so that its tail 2 enters a slot in the driver faceplate 3. The faceplate is screwed on the spindle nose.

Often a driver chuck with a safety dog is employed instead of an ordinary faceplate.

When very long workpieces ( $\frac{L}{D} > 10$  or 12) are turned between centres, rests are used to prevent sag in the middle of the work.

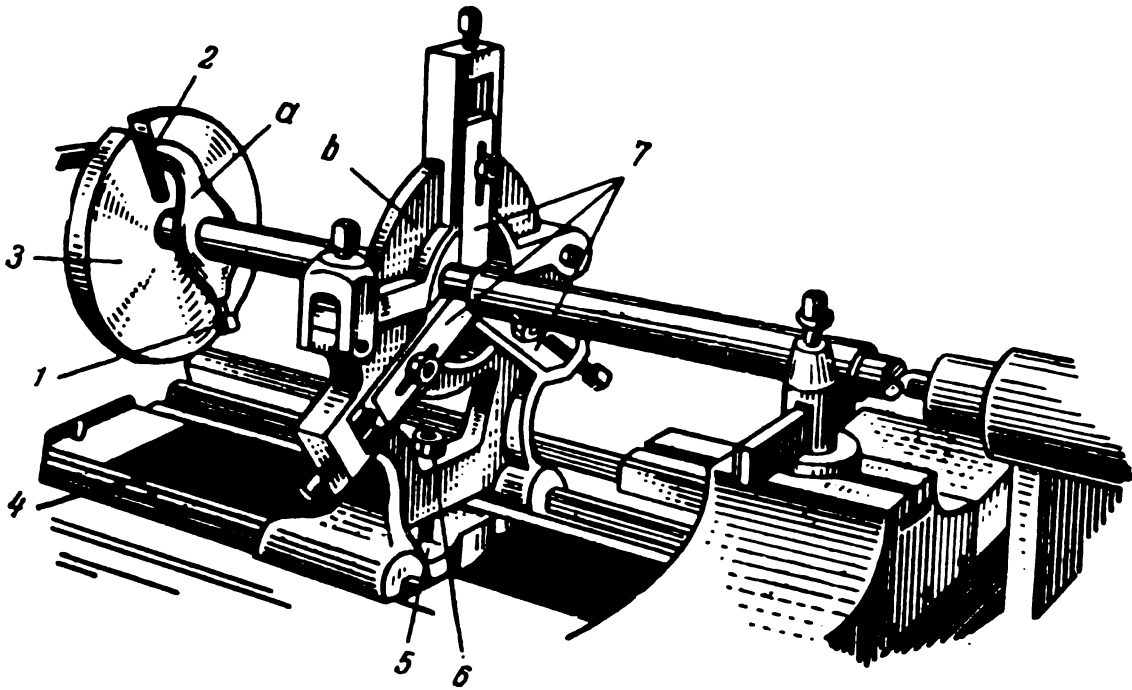


Fig. 203. Turning between centres with the work supported by a steady rest

A *steady rest b* (see Fig. 203) is mounted on both ways 4 of the lathe bed where it is clamped by the base clamp 5, bolt and nut 6 between the headstock and tailstock. The workpiece is supported by three adjustable jaws 7. Steady rests designed for high-velocity turning have ball or roller bearings contacting the rotating workpiece, the bearings being built into the jaws.

A *follower rest* performs the same functions as the steady rest but it is mounted on the saddle and moves together with the tool. It has two jaws which support the work opposite the tool.

**Turning work held in a chuck.** Workpieces with a length  $L < 4D$  may be turned with straight-turning, facing, cut-off or boring tools. Such workpieces can be clamped in a chuck without the need of additional support of the free end by the dead centre or by a steady rest. Three- and four-jaw chucks, screwed on the spindle nose, are employed for this purpose.

A three-jaw universal (self-centring) chuck (Fig. 204a) is used for holding symmetrical work as a rule. Here the three jaws 1, 2

and 3 move together towards or away from the centre of the chuck due to the action of the large bevel gear 4 on the back of which a spiral scroll has been cut. The bevel gear 4 is rotated by any one of

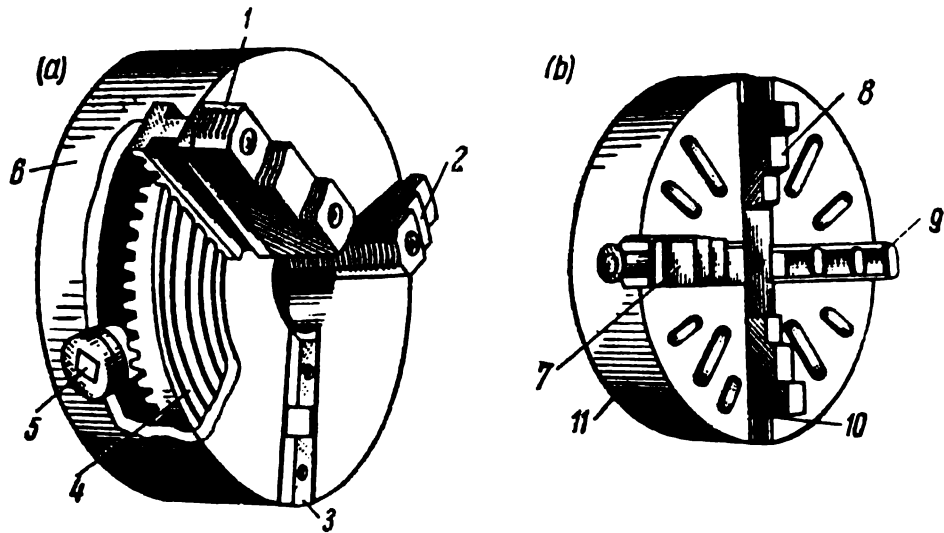


Fig. 204. Lathe chucks:  
(a) three-jaw universal chuck; (b) four-jaw independent chuck

three bevel pinions 5 mounted in the chuck body 6 and turned by inserting a chuck wrench into their square sockets. Projections, or teeth, on the bottom of the three jaws engage the spiral grooves of the scroll. Upon rotation of the large bevel gear the jaws move simultaneously along radial slots in the chuck body and clamp the work.

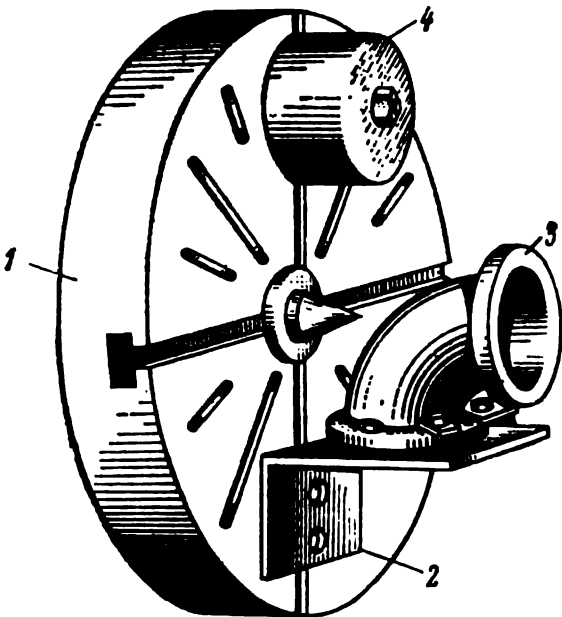


Fig. 205. Clamping work on a faceplate

In the four-jaw independent chuck (Fig. 204b) each of the four jaws 7, 8, 9 and 10 is moved by its own screw independent of the other jaws along radial slots of the chuck body 11. Chucks of this type find application in turning work of complex and asymmetrical shapes.

Thin plates and other thin flat work can be conveniently faced if they are held by magnetic chucks with permanent magnets.

Pneumatically and hydraulically operated chucks are used to speed up and facilitate handling operations of certain parts in mass and large-lot production.

**Turning work on a faceplate.** A faceplate 1 (Fig. 205) screwed on the spindle nose may prove to be highly efficient in machining asymmetrical work or work of complex and irregular shape which is incon-

venient or even impossible to clamp in jaw-type chucks. A faceplate usually has four T-slots and a number of plain radial slots.

The workpiece is clamped to the faceplate with bolts and straps. Sometimes it is more convenient to mount an angle plate 2 on the faceplate and to clamp the work 3 on the angle plate. The work and angle plate can be counterbalanced by a counterweight 4 mounted opposite on the faceplate.

**Boring in a lathe.** This lathe operation is done inside a hole by means of a boring tool (see Fig. 202e); the work is clamped in a chuck or on a faceplate. In some cases a hole is drilled from the solid before boring.

**Facing.** Either the whole end surface of the workpiece or an annular end surface, such as a shoulder, can be faced with cross feed of the tool (Fig. 202c). Facing tools are employed for this operation, the work being held in a chuck or on a faceplate. Facing may be done either from the periphery of the work towards the centre or from the centre outwards.

**Cutting off and cutting grooves.** For these operations the work is held in a chuck, and cross feed of the tool is used. Special grooving tools are employed when grooves of special shapes are cut.

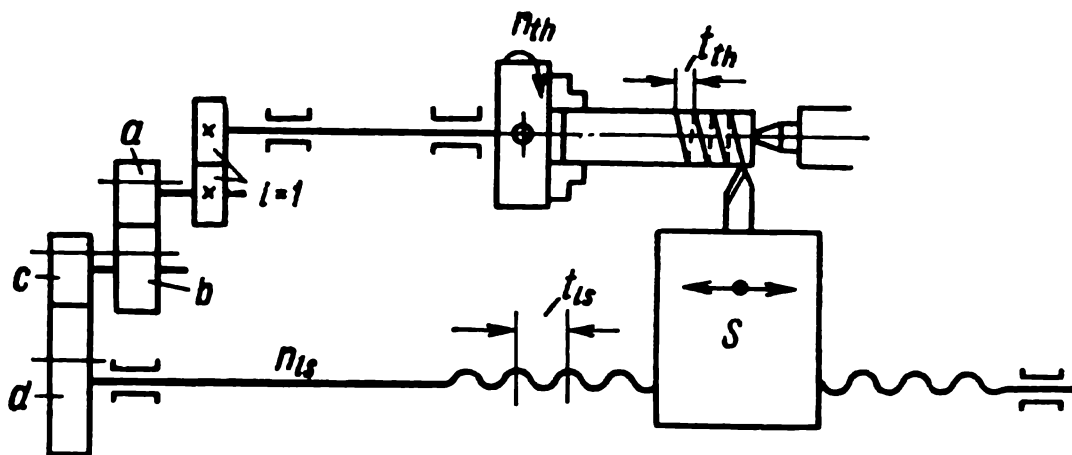


Fig. 206. Principle of thread cutting in a lathe

**Cutting screw threads in a lathe.** In cutting screw threads on engine lathes that do not have a quick-change gearbox or when the thread pitch is nonstandard and not accommodated by the given gearbox or when precise threads are required, change gears must first be selected and set up on the quadrant as shown in Fig. 206. As a rule, the gearing ratios of the pair of gears linking the spindle to the reversing mechanism and that of the mechanism itself are equal to unity.

The equation of the spindle-to-carriage gear train in cutting screw threads is deduced from the condition that in one revolution of the



spindle, carriage travel must equal the pitch  $t_{th}$  of the screw thread to be cut; therefore

$$t_{th} = t_{ls} i_{cg}, \quad (112)$$

or

$$\frac{t_{th}}{t_{ls}} = i_{cg}, \quad (113)$$

where  $t_{ls}$  = pitch of the lead screw, in mm,

$i_{cg}$  = gearing ratio of the change-gear quadrant.

For quadrants with two pairs of gears the set-up formula for change gear selection in cutting screw threads will be

$$\frac{t_{th}}{t_{ls}} = i_{cg} = \frac{a}{b} \times \frac{c}{d}, \quad (114)$$

where  $a$ ,  $b$ ,  $c$  and  $d$  = numbers of teeth of the change gears.

The equation obtained determines the kinematic linkage between the lead screw and the lathe spindle. Various thread cutting problems can be solved by the use of this equation to select gears with the required number of teeth to be set up on the quadrant. The values of  $t_{th}$  and  $t_{ls}$  may be expressed in inches, millimetres, threads per inch, modules, etc. However, the values substituted into the formula must be in the same system of units or else appropriate conversion factors must be introduced.

In thread cutting practice a thread pitch is said to be even if the lead screw pitch is an exact multiple of this pitch and odd if it is not.

An even screw thread can be cut by disengaging the half nuts at the end of each pass, retracting the tool and returning the carriage back to the beginning of the thread manually and without stopping or reversing the lathe. When the half nuts are engaged for the next pass and the tool is advanced again it will exactly follow the previously cut thread groove.

This procedure cannot be followed in cutting an odd thread. Here, the half nuts are not disengaged after each pass. The lathe is stopped at the end of the pass, the tool is withdrawn and the carriage is returned to the beginning of the thread by reversing the lathe, otherwise the tool will not follow the previously cut thread groove in the subsequent pass.

In thread cutting, the work is mounted between centres or clamped in a chuck, depending upon its length. The form of the threading tool must coincide with the profile of the thread being cut.

**Turning tapers.** Tapered surfaces can be turned by employing one of the following methods: by swivelling the compound rest to the required angle, by setting over the tailstock, by using a broad-nose tool and by using a taper attachment.

The angle  $\alpha$  through which the compound rest must be swivelled to turn a specified taper (Fig. 207a) is determined from the formula

$$\tan \alpha = \frac{D-d}{2l}, \quad (115)$$

where  $D$  = larger diameter of the taper,

$d$  = smaller diameter,

$l$  = length of the taper.

After swivelling the rest to this angle about axis  $l$  it is clamped in this position.

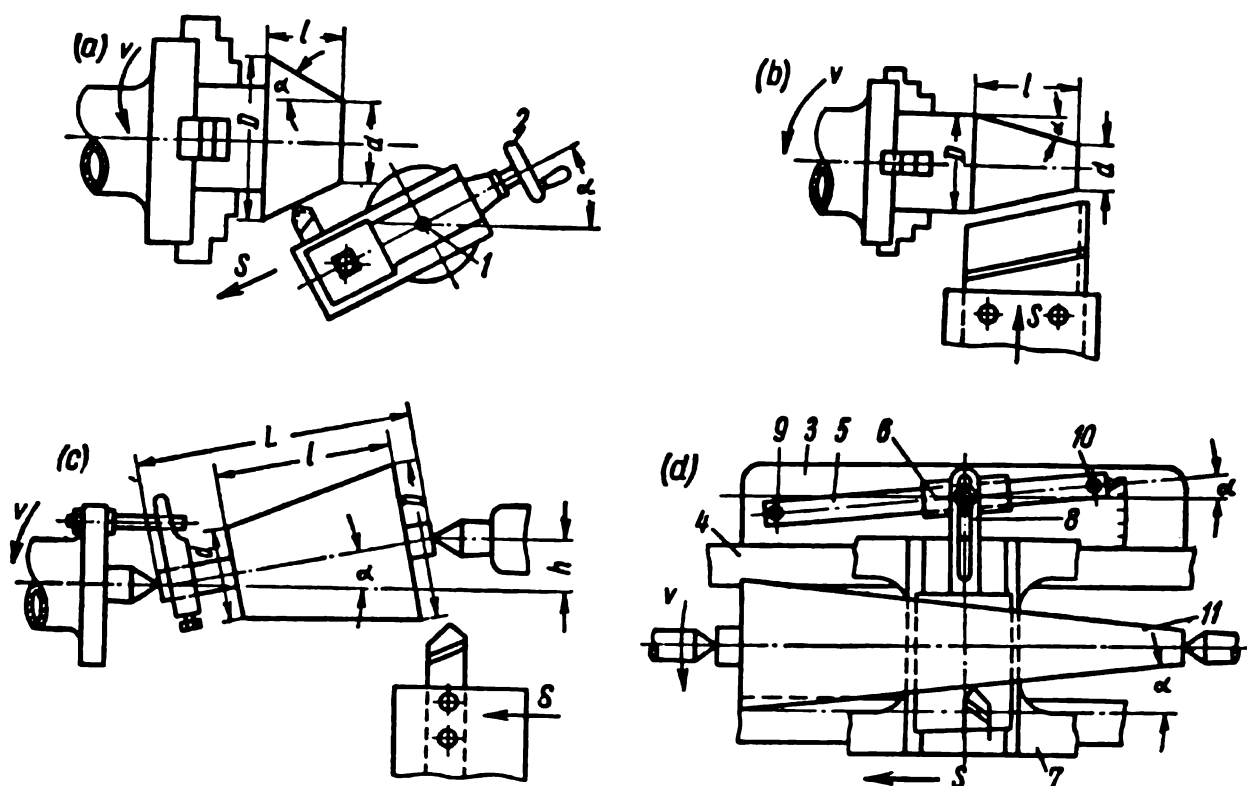


Fig. 207. Taper turning methods:

(a) by swivelling the compound rest; (b) with a broad-nose tool; (c) by setting over the tailstock; (d) by using a taper attachment

The taper is turned with hand feed by rotating handle 2. This procedure can be employed for turning short internal and external tapers with a large angle of taper.

The work is commonly held in a chuck and turned with a straight-turning tool.

Short external tapers with various angles of taper can also be turned with a broad-nose tool (Fig. 207b) using cross feed. The width of the tool slightly exceeds that of the taper being turned; the work is held in a chuck or it is clamped on a faceplate.

Long workpieces with a small angle of taper ( $\alpha$  not exceeding  $8^\circ$ ) are usually turned by setting over the tailstock centre (Fig. 207c) by the amount  $h$  which can be determined approximately from the

formula

$$h = \frac{L(D-d)}{2l} \text{ mm,} \quad (116)$$

where  $L$  = full length of the workpiece, in mm.

When the tailstock has been set over by the amount  $h$  (usually within 15 or 20 mm) the element of the taper being turned will be parallel to the direction of longitudinal tool feed so that power feeds can be used for the operation.

Long tapered work is frequently turned with a taper attachment (Fig. 207*d*). For this purpose guide bar 5 with slide 6 is mounted on bracket 3 which is fastened to lathe bed 4. The cross slide 7 of

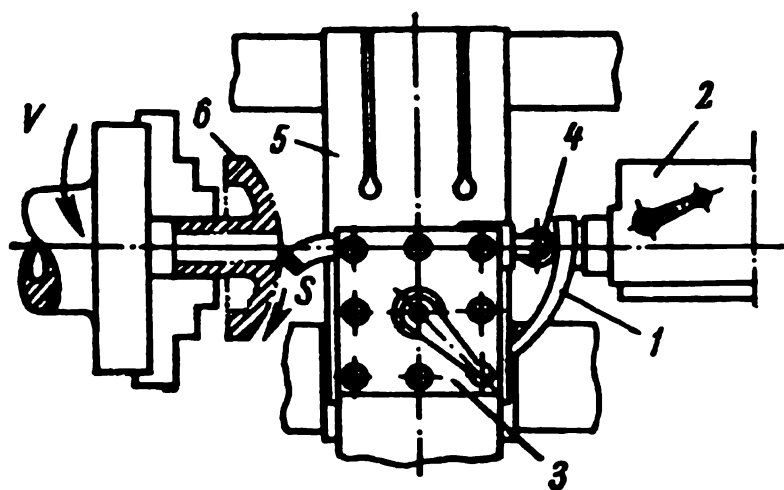


Fig. 208. Turning a contoured surface with a tracer device

the lathe is disengaged from the cross feed screw and is linked through tie 8 to the slide 6. Guide bar 5 is clamped by bolts 9 and 10 at an angle  $\alpha$  determined by formula (115).

As the carriage travels in a longitudinal direction, slide 6 follows along the guide bar and moves the tool in a direction square to the axis of workpiece 11. As a result of the combination of these two motions tool travel is parallel to the guide bar and the work is turned to the specified taper.

**Form turning operations.** Contoured surfaces of revolution may be turned on a lathe with form tools or by means of a tracer device.

The contour of the form tool must strictly comply with that of the workpiece.

Form tools are usually employed only for finishing and therefore they can be used for a long time and retain their contour well.

Tracer devices applied for turning contoured surfaces may be either mechanically or hydraulically operated.

The mechanical tracer device shown in Fig. 208 was devised by lathe operator A. Ionov. Template 1 is mounted in the spindle of

tailstock 2. Roller 4 is clamped in the square turret 3 opposite the single-point tool and is in contact with template 1. The saddle of carriage 5 is free to move in a longitudinal direction. Upon cross feed of the tool and constant contact between roller 4 and template 1, a spherical surface will be turned on workpiece 6. Contoured surfaces of other shapes may be turned with this device with cross feed by merely changing the template to one with the required contour.

The tracer device used to turn contoured surfaces with longitudinal feed is the same as the taper attachment except that a template with the required contour is substituted for the guide bar. A hydraulic tracer slide is often installed on an engine lathe when a great deal of contour turning is to be done with longitudinal feeds.

## Chapter 33

### DRILLING PRACTICE

#### 33-1. The Drilling Process

*Drilling* is an extensively used process by which through or blind holes are originated or enlarged in a workpiece. Various types of drills serve as tools for this purpose.

*The cutting speed  $v$*  in drilling is the peripheral speed of a point on the cutting edge of the drill located at the maximum distance from the drill axis.

The selection of the cutting speed in drilling depends upon a number of factors—the properties of the material being drilled and of the drill material, drill diameter, rate of feed, cooling facilities, etc. For example, a drill tipped with cemented carbide, grade BK8, can drill steel at speeds ranging from 45 to 80 m per min and cast iron at 50 to 95 m per min.

*The depth of cut  $t$*  in drilling from the solid is equal to one half of the drill diameter (Fig. 209).

*The feed  $s$*  is the movement of the drill along its axis in millimetres per revolution.

*The undeformed-chip cross-sectional area* (chip cross section) is twice the product of the chip thickness  $a$  and the chip width  $b$  (both thickness and width refer to the undeformed chip).

*The machining time* in drilling from the solid is calculated on the basis of the length of the cut, rate of feed and drill speed.

The power  $N_c$  required in drilling is determined from the cutting resistance torque  $M_t$  and the drill speed  $n$  in rpm:

$$N_c = \frac{M_t n}{716.2 \times 1000 \times 1.36} \text{ kW}, \quad (117)$$

where 716.2=factor for converting kg-m per sec into metric hp in rotary motion.

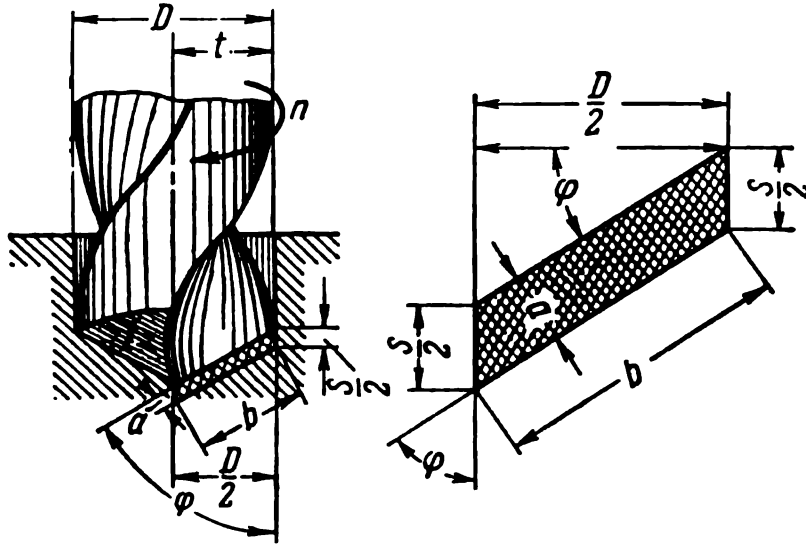


Fig. 209. Elements of the cutting process in drilling

### 33-2. Cutting Tools for Drilling and Machining Holes

The most widely employed drilling tools are twist and flat drills.

The *twist drill* (Fig. 210a) consists of the body and point, neck and shank. The point comprises the cutting elements while the body guides the drill in operation.

The body has two helical flutes 1 and 2 for assisting in removing chips out of the drilled hole, and two margins 3 and 4 which reduce the friction between the drill and hole wall, facilitate the removal of heat generated in drilling and properly guide and locate the drill.

The drill point has two cutting edges, or lips, 5 and 6, a web, or chisel edge, 7 and two relief surfaces 8.

The point angle  $2\phi$  of a drill, formed by the lips, is selected to suit the hardness and brittleness of the material being drilled. A point angle of  $116^\circ$  to  $118^\circ$  is used for medium-hard steel and cast iron,  $125^\circ$  for hardened steel and  $130^\circ$  to  $140^\circ$  for brass and bronze. The helix angle  $\epsilon$  of the flutes ranges from  $24^\circ$  to  $30^\circ$  for most drills.

Twist drills are manufactured of high-quality tool and high-speed steels and also in carbide-tipped designs (with a cemented carbide tip brazed into the point to form the lips). Twist drills are available in a size range from 0.25 to 80 mm.

**Flat drills** (Fig. 210b) consist of the head, neck and shank. The head may have either parallel sides or sides with a certain amount of back taper. Those with parallel sides are more accurate in operation and have a higher tool life. A back taper of  $2^\circ$  or  $3^\circ$  may be provided to reduce the friction in drilling. The point angle  $\phi$  of a flat drill has a value from  $90^\circ$  to  $120^\circ$ .

Holes obtained by drilling, cored holes in castings and holes punched in forgings are further machined with core drills and reamers.

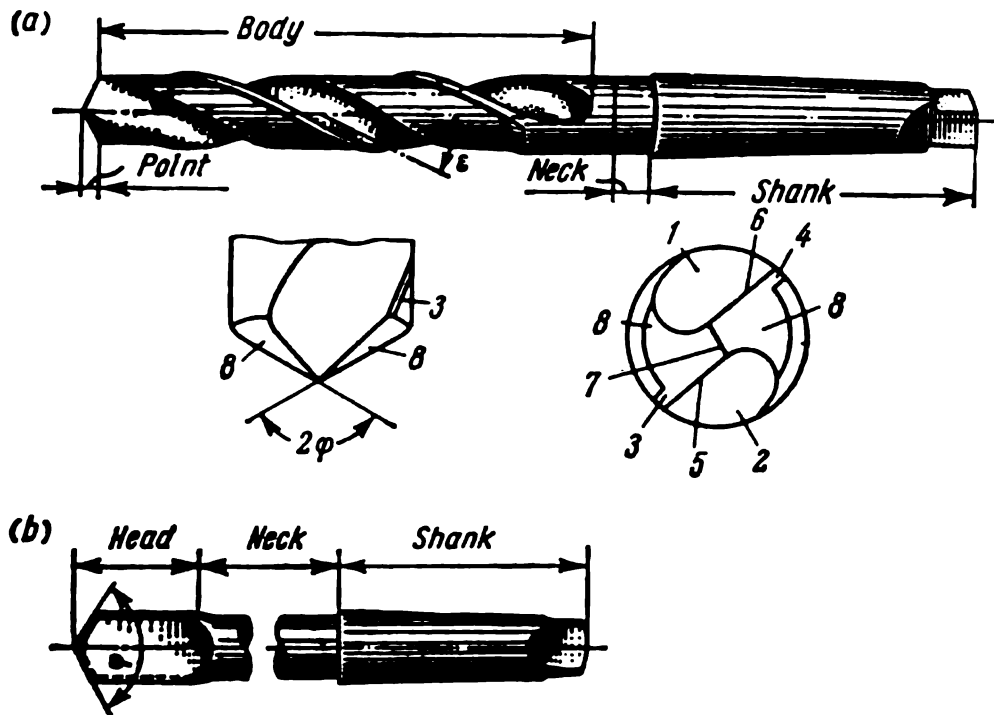


Fig. 210. Drills:  
(a) twist drill, its parts and elements; (b) flat drill

Core drills produce holes of the 3rd to 5th grades of accuracy and with a surface quality of the 4th or 5th classes. In reaming, holes of the 2nd or 3rd grades of accuracy may be obtained with a finish of the 7th or 8th classes.

Three- and four-flute core drills and reamers of various types are employed for these operations.

A **core drill** (Fig. 211) consists of the chamfer, body, neck and shank.

The cutting elements of a core drill consist of the flat end, or web 1, and the chamfer with three or four cutting edges, or lips, 2. Each lip is formed by the face 3 and relief surface 4. The lip angle  $2\phi$  may vary from  $90^\circ$  to  $120^\circ$ .

The body or guiding part of a core drill has three or four helical flutes and as many margins 5. The latter are the actual guiding and locating elements of the core drill and ensure high machining accu-

racy by preventing deviation of the tool to one side in operation. The flutes are usually designed with a helix angle  $\epsilon$  of from  $10^\circ$  to  $30^\circ$ .

In addition to the shank type, core drills may be of the shell and carbide tipped types.

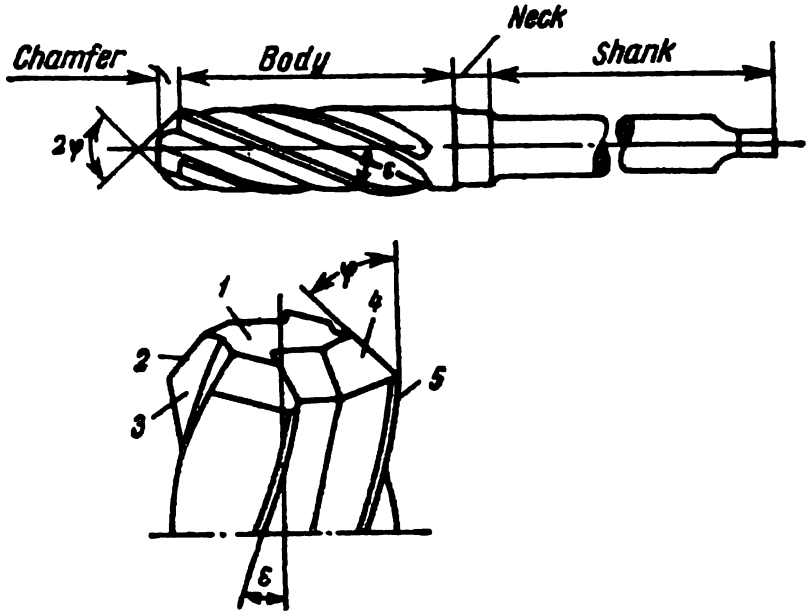


Fig. 211. A core drill, its parts and elements

A *reamer* (Fig. 212a) has a fluted section, neck and shank. The fluted section includes the chamfer  $l_1$ , starting taper  $l_2$ , sizing section  $l_3$  and the back taper  $l_4$ . The main cutting action in reaming is

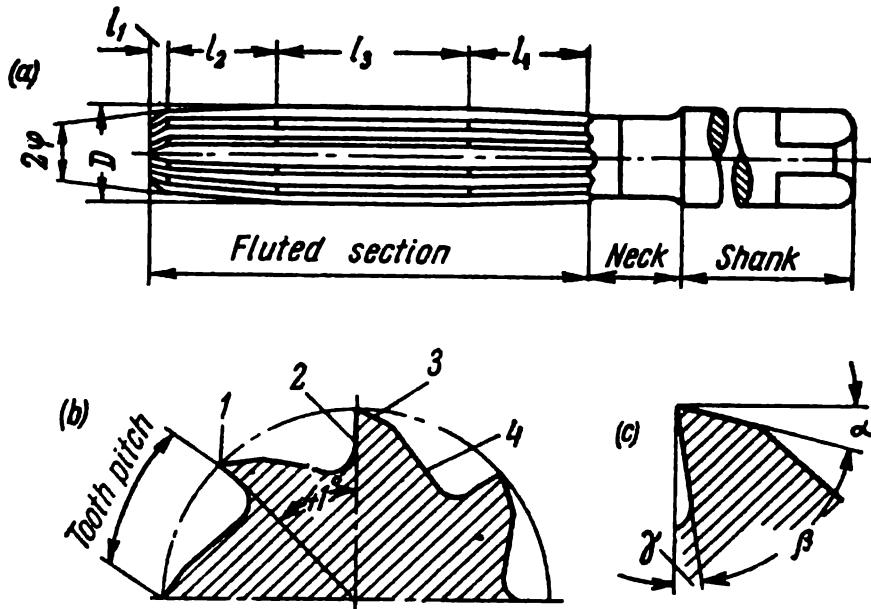


Fig. 212. A reamer, its parts and elements

performed by the starting taper  $l_2$ , each tooth of which has a main cutting edge 1, face 2, relieved land 3 and flute surface 4 (Fig. 212b). The face is the surface along which the chips slide as they are cut

from the work; the relieved land reduces the friction between the tooth and the reamed surface. As in single-point tools, reamer teeth have three angles—rake angle  $\gamma$ , relief angle  $\alpha$  and lip angle  $\beta$  (Fig. 212c).

The chamfer, or bevel,  $l_1$  is ground off to an angle  $2\phi$  which is equal to from  $1^\circ$  to  $3^\circ$  for a hand reamer, from  $8^\circ$  to  $10^\circ$  for a machine reamer for reaming steel and from  $20^\circ$  to  $30^\circ$  for a machine reamer for cast iron. The chamfer angle  $2\phi$  affects the axial force required in reaming; the larger the angle, the larger the force required.

The sizing section  $l_2$  serves to guide the reamer and, to some extent, it smooths (sizes) the hole.

The back taper  $l_3$  following the sizing section (with a difference between the maximum and minimum diameters of from 0.01 to 0.08 mm) reduces friction between the reamer and the hole surface.

According to the hole to be reamed reamers may be either of the cylindrical or taper type. Reamers are frequently tipped with cemented carbides to increase their production capacity.

### 33-3. Drilling Machines

Drilling machines are classified as:

- (1) general-purpose machines—single- and multiple-spindle upright drilling machines; radial and bench-type drilling machines;
- (2) specialised machines—unit-type, multiple-spindle, gang or straight-line type, automatic drilling machines and others;
- (3) single-purpose machines—centre-drilling and deep-hole drilling machines (horizontal and vertical types); semi-automatic and automatic drilling machines.

Boring machines, which resemble drilling machines in many respects, can be classified as: horizontal boring, drilling and milling machines, vertical boring machines, precision or single-point boring machines, jig boring machines, semiautomatic boring machines, etc.

Fig. 213a illustrates a single-spindle upright drilling machine. Here the box column 2 is bolted to the base 1. The motor 3, mounted at the top of the column, powers the drill 4 inserted into the lower end of the spindle 5. The rotary motion of the drill is the primary cutting motion. This motion is transmitted to the drill through the speed gearbox 6 in the overhanging housing at the top of the machine. The drill obtains its axial feed motion from the feed gearbox mounted in the drill head 7 which can be adjusted vertically along the column.

Vertical movement of the spindle and drill may be accomplished either by power from the feed gearbox according to a preset rate of feed or manually from handwheel 8. Depending upon its shape and size the work can be clamped on table 9 in a machine vise, drilling



jigs or other holding fixtures. Turning crank *10* will either raise or lower the table to the most convenient height for each job.

Single-spindle upright drilling machines can drill holes up to 75 mm in diameter and up to 350 mm deep. These machines have

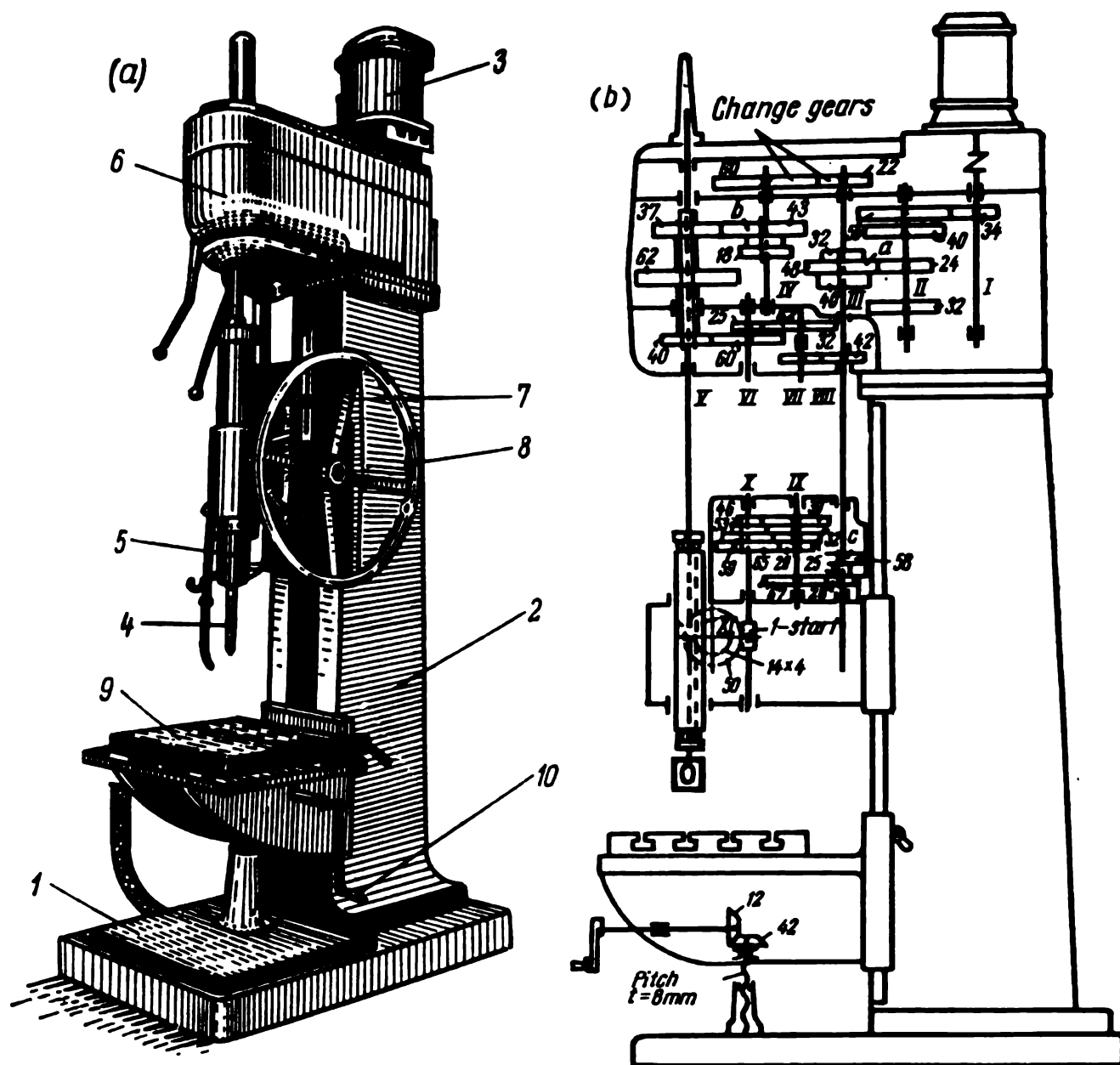


Fig. 213. Single-spindle upright drilling machine, model 2135:

(a) external view; (b) gearing diagram

a wide range of spindle speeds and feeds and are employed, therefore, not only for drilling from the solid, but also for core drilling, reaming and tapping operations.

The most expedient cutting speed in drilling and other hole making operations is attained by selecting the corresponding spindle speed. This is done by changing the gearing ratio  $i_{sg}$  of the speed gearbox where either sliding cluster gear *a* is shifted as required on

shaft *III* (Fig. 213*b*) or cluster gear *b* on shaft *IV*. The various positions of the two cluster gears provide six spindle speeds in a range from 47 to 466 rpm. If the speed  $n_m$  of the motor is known the equation of the spindle drive gear train can be expressed as follows:

$$n_{sp} = n_m i_{sg} \text{ rpm,} \quad (118)$$

where  $i_{sg}$  = gearing ratio of the speed gearbox,  
 $n_{sp}$  = spindle speed, in rpm.

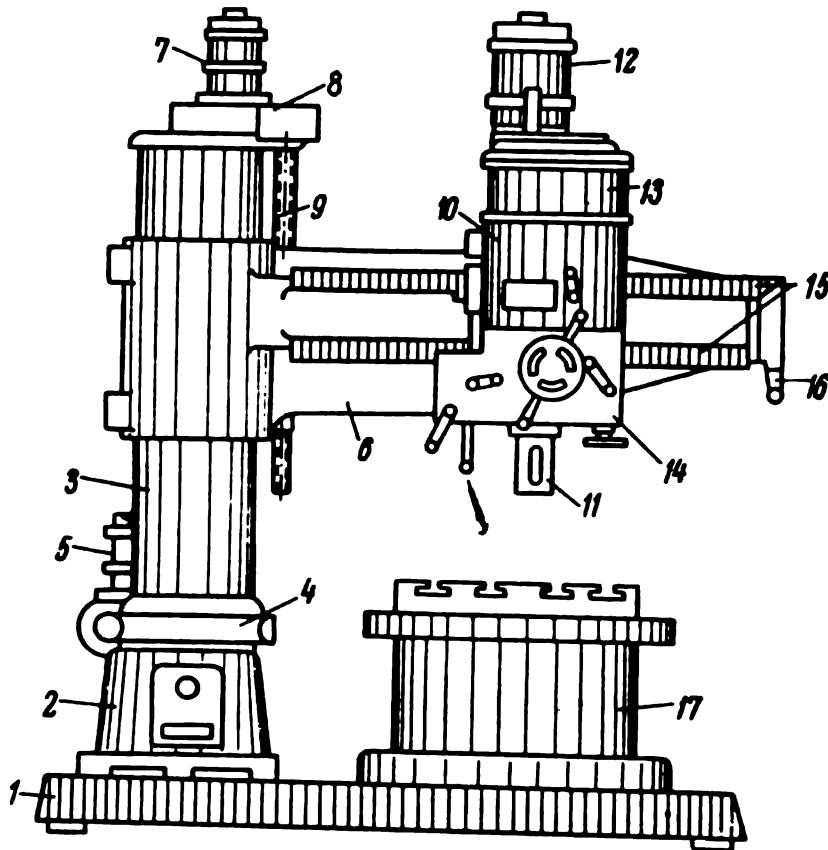


Fig. 214. Radial drilling machine, model 2B56

The desired drill feed is selected by shifting the sliding cluster gear *c* in the feed gearbox (Fig. 213*b*) and shifting the sliding key of the intermeshing gear cones on shafts *X* and *IX*. Rotation is transmitted further through worm gearing to a rack pinion with 14 teeth which meshes with a rack (module—4 mm) secured to a sleeve in which the spindle freely rotates. The spindle is fed in the axial (vertical) direction together with the sleeve.

The model 2135 upright drilling machine has eight rates of feed ranging from 0.1 to 1.11 mm per revolution.

A radial drilling machine is shown in Fig. 214. These machines are available in an extensive size range and are used for drilling operations in heavy and bulky workpieces which are either inconvenient or impossible to mount on the table of an upright drilling machine.

Footing 2 with column 3 which bears the main forces acting in the machine are secured to the base 1. The split yoke 4, clamping the column to the footing, is actuated by a special device which is powered from motor 5. Arm 6 can be raised or lowered on the column and clamped at the required position; vertical adjustment of the arm is powered from motor 7 through two-stage reducing gear 8 and elevating screw 9. Drill head 10 which can move along the arm transmits the preset primary (rotary) cutting motion and the axial feed motion to the spindle from motor 12 through speed gearbox 13 and feed gearbox 14.

The model 2B56 (Soviet) radial drilling machine has 12 different spindle speeds and 9 different feeds ranging from 55 to 1440 rpm and from 0.19 to 1.9 mm per revolution, respectively.

If two change gears provided in the gearbox are interchanged, the number of different spindle speeds can be doubled.

Drill head 10 can be clamped on the arm to provide rigidity during drilling; it operates as an independent unit and can be traversed along the racks 15 of the arm in a radial direction by means of a special lever. The arm is swung manually about the column axis with lever 16 after releasing the column clamping yoke 4.

In drilling on the radial drilling machine the workpiece is clamped either on the removable table 17 or directly on the base 1 while the spindle with the drill is set to the working position by three movements: vertically along the spindle axis, radially along the arm and by swinging the arm about the column axis through a definite angle.

### 33-4. Principal Types of Drilling Operations

The two main types of operations done on drilling machines are drilling from the solid and enlarging holes. Besides these, core drilling, countersinking, counterboring, spot facing, reaming, tapping and other operations can be done.

**Drilling.** This operation may be divided into ordinary and deep-hole drilling.

In *ordinary drilling* twist drills with straight or taper shanks are commonly used to produce through or blind holes. Taper shank drills are inserted directly into the taper hole of the drilling spindle when the sizes of the tool and spindle hole tapers coincide; otherwise adapting devices such as sockets or sleeves (Fig. 215a) are employed. Straight shank drills are clamped with two- or three-jaw drill chucks (Fig. 215b) either by hand or by means of a chuck wrench, depending upon the design of the chuck.

The workpiece is located and clamped on the work table in a machine vise, V-blocks for round work, plate, box or pump jigs, etc. The type of work holding device used depends upon the shape and size of the workpiece, the required accuracy and the rate of production.

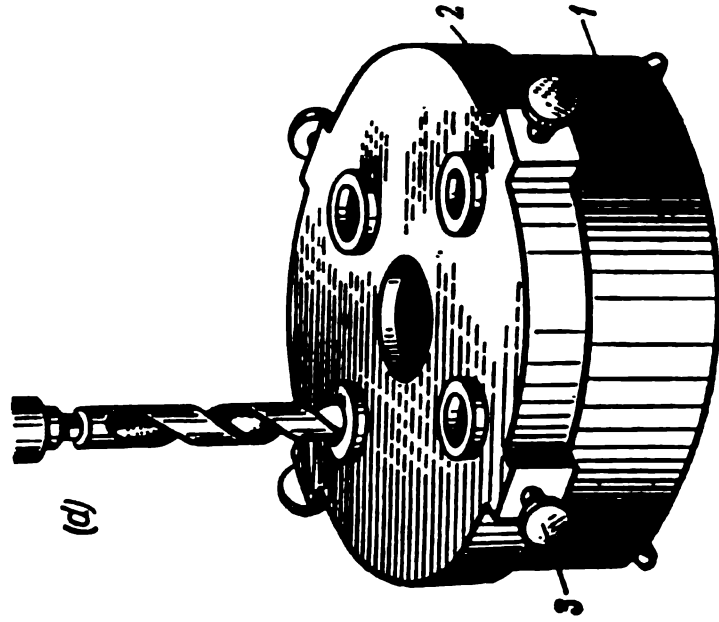
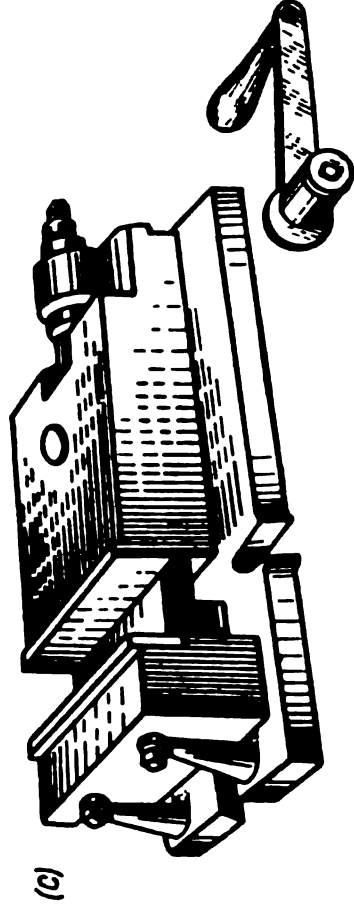
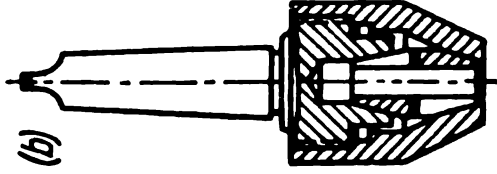
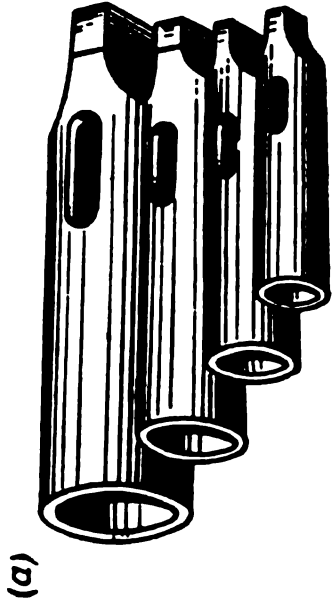


Fig. 215. Tool and work holding devices for drilling machines:  
(a) drill sleeves; (b) chuck for straight-shank drills; (c) machine vise;  
(d) plate jig

Most commonly used are machine vises (Fig. 215c) and drill jigs which are special devices designed to hold a particular workpiece and to guide the cutting tools. Jigs enable drilling to be done without previously laying out the workpiece. When a plate jig (Fig. 215d) is used the workpiece 1 is clamped either in a vise or directly on the work table. The plate jig 2 is mounted on the surface of the workpiece where the holes are to be drilled and clamped with screws 3.

The specified depth is achieved in drilling blind holes by using the depth gauges furnished on certain drilling machines or by fitting a stop sleeve directly on the drill.

In *deep-hole drilling* where the length-to-diameter ratio of the hole is 10 or more, the following special types of drills are used:

- (1) Gun drills—for drilling holes up to 25 mm in diameter.
- (2) Half-round drills—for holes over 25 mm in diameter.
- (3) Trepanning drills—for annular drilling of holes over 80 mm in diameter (the drill cuts an annular groove through the work leaving a core which enters the drill during operation).

Deep-hole drilling machines are most frequently of the horizontal type; the workpiece is rotated by a chuck and supported by a steady rest while the drill is fed axially. This procedure reduces the amount by which the drill departs from the hole axis in drilling.

**Enlarging holes.** This process may be applied for further machining of a previously drilled hole. It is used, as a rule, in drilling holes over 30 mm in diameter. Large twist drills have a web of considerable thickness requiring extremely high feed forces when they are used to drill a hole from the solid. Moreover, such large drills often deviate from the true drilling axis. It proves more expedient, in many cases, to first drill a hole less than the specified diameter (from 0.2 to 0.4 of the required diameter) and then to enlarge it with a drill of the required size.

## Chapter 34

### MILLING PRACTICE

#### 34-1. The Milling Process

*Milling* is a machining process for producing flat, contoured and helical surfaces; for cutting threads and toothed gears and for making helical grooves by means of rotating multiple-edged tools called *milling cutters*.

Milling involves simultaneous rotary motion of the cutter and, usually, linear feed motion of the work.

Milling may be done, insofar as the directions of cutter rotation and workpiece feed are concerned, by either of two methods: (1) up, or conventional, milling in which the work is fed against cutter rotation and (2) down, or climb, milling in which the cutter rotation and work feed directions coincide.

In *conventional milling* the load on each tooth of the cutter gradually increases and reaches its maximum value as the tooth leaves the cut. This ensures smoother operation of the milling machine and less tool wear if castings with a chilled skin or forgings with scale are milled. Not a very high quality of surface finish is attained, however, and therefore this method is used in roughing.

In *climb milling* the cutter tooth cuts a chip of maximum thickness at the beginning as it enters the cut so that it is subject to maximum load. This method can never be used in a machine unless it is equipped with a backlash eliminator on the feed screw. The forces are directed downwards in milling by this method (the opposite being true for conventional milling); this facilitates the clamping of workpieces that cannot be easily held. Another advantage is the better finish obtained.

The *cutting speed*  $v$  in milling is the peripheral speed of the cutter measured at its outside diameter.

Cutting speed selection depends upon the properties of the material being milled, cutter material, diameter and life of the cutter, feed, depth and width of cut, as well as the number of cutter teeth, cooling facilities, etc.

The *feed*  $s$  in milling is the movement of the work relative to the cutter axis in a unit of time (Fig. 216). It may be expressed as feed per tooth (mm/tooth), feed per cutter revolution (mm/rev) or feed per minute (mm/min).

The heaviest feasible feed is employed for rough milling; thus, for high-speed steel plain milling cutters the feed may be 0.05 to 0.6 mm/tooth for milling steel and 0.1 to 0.8 mm/tooth for cast iron.

In semifinish and finish milling the rate of feed is limited by the specified surface finish that must be maintained.

The *depth of cut*  $t$  is the thickness of the layer of metal removed in one cut (Fig. 216). A depth of cut from 3 to 8 mm is common in roughing operations and from 0.5 to 1.5 mm in finishing.

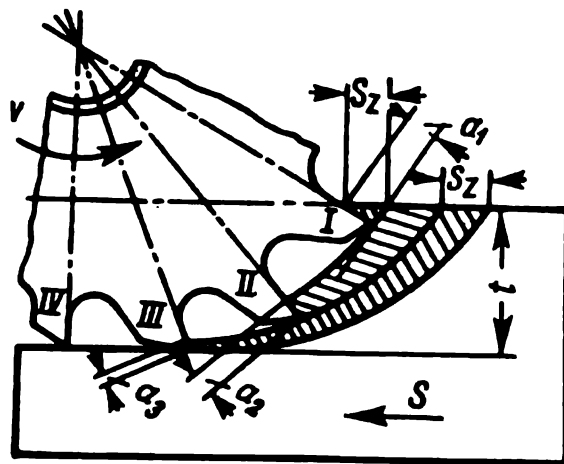


Fig. 216. Feed in milling and the thickness of the chip cut by the cutter teeth

*The width of cut  $B$*  is the width of the work surface contacting the cutter in a direction perpendicular to the feed.

*The cross-sectional area of the chip* removed by one tooth of the cutter (for instance, a plain cutter) is the product of the width of cut  $B$  and the chip thickness  $a$ :

$$f = Ba \text{ mm}^2. \quad (119)$$

The chip thickness is a variable value; it varies from zero at the moment the tooth enters the cut (conventional milling) to the maximum value  $a_1$  at tooth exit (see Fig. 216).

Since not one, but several teeth are in contact with the work simultaneously, it is necessary to deal with the total cross-sectional area of the undeformed chip removed by several teeth. For the case illustrated in Fig. 216 the thickness of the undeformed chip is  $a_1$  for the first tooth,  $a_2$  for the second,  $a_3$  for the third, while for the fourth tooth  $a_4=0$ .

The total cross-sectional area of the undeformed chips being removed at any instant by all the teeth in the cut has a direct effect on cutting forces in milling.

*Machining time* in milling is the time required for the process of metal cutting in one pass of the cutter. In calculating the machining time it is necessary to take into consideration the total length of table (or cutter) travel in the direction of the feed (including cutter approach and overtravel), the rate of feed and the number of passes.

### 34-2. Milling Cutters

Milling cutters of a great many varieties are the cutting tools employed for milling.

A milling cutter is a multiple-edged tool having the shape of a solid of revolution with cutting teeth arranged either on the periphery or on end face or on both.

Milling cutters may be classified as plain and face milling cutters for machining flat surfaces; side and angle cutters and end mills for slots, grooves, flutes and splines; formed milling cutters for contoured surfaces; gear tooth cutters for cutting gears; gear hobs for cutting spur, helical and worm gears by the generating method, etc.

A tooth 1 (Fig. 217a) of a cutter is, in principle, a single-point tool having a main cutting edge 2, face 3, land 4 and back of tooth 5.

*The chief angles* of the cutter,  $\gamma$ ,  $\alpha$ ,  $\beta$  and  $\delta$ , are measured in the cross section  $N-N$  perpendicular to the cutting edge.

Angle  $\gamma$  is called the radial rake; it facilitates removal of the chips being cut and reduces their contraction. The radial rake usually ranges from  $10^\circ$  to  $20^\circ$ , larger values being used for milling soft ma-

materials and smaller values for harder materials. Carbide tipped cutters have a negative rake angle  $\gamma$  with a value from  $-10^\circ$  to  $-15^\circ$ .

Angle  $\alpha$  is called the relief angle; it reduces the friction between the tooth land and the cutting surface. The relief angle varies from  $12^\circ$  to  $25^\circ$  for various types of cutters.

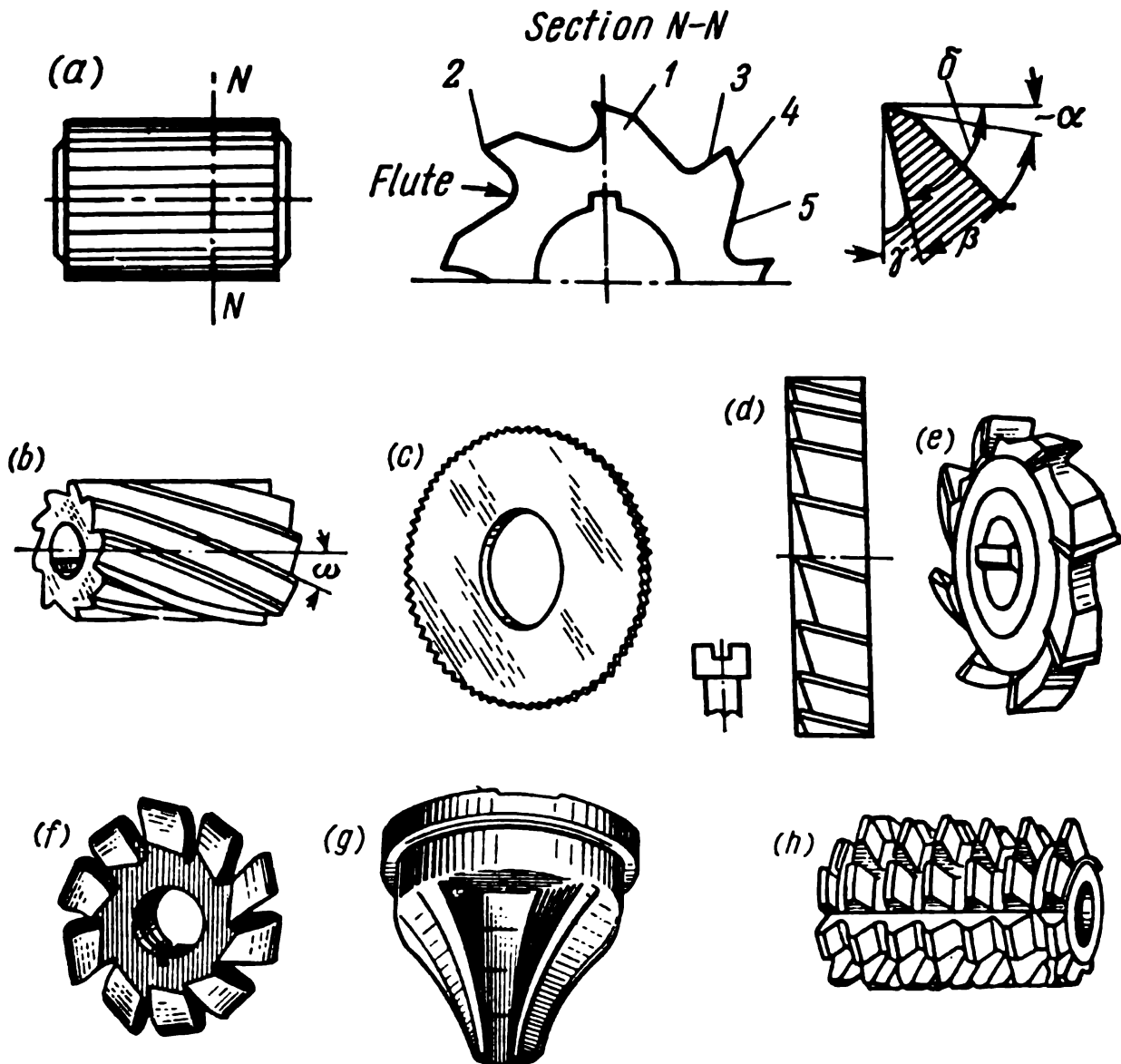


Fig. 217. Principal types of milling cutters:

(a) straight-flute plain milling cutter; (b) helical plain milling cutter; (c) plain metal slitting saw; (d) half-side milling cutter; (e) side milling cutter; (f) gear milling cutter; (g) end-mill type gear milling cutter; (h) gear hob

A plain milling cutter may have either straight or helical teeth in which the latter have a helix angle of  $\omega$  (Fig. 217b). Standard helical plain milling cutters have a helix angle  $\omega = 20^\circ$  to  $30^\circ$ ; side cutters and end mills have a helix angle from  $10^\circ$  to  $15^\circ$ .

In the operation of a cutter with straight teeth each tooth enters the cut simultaneously along its full length. This leads to intermittent



loads acting on the machine and impairs the quality of the surface milled.

Cutters with helical teeth operate more smoothly since the teeth enter the cut gradually and the load on the machine is more uniform.

### 34-3. Milling Machines

*Ordinary* milling machines include the plain horizontal and the vertical spindle types, thus named because of the horizontal and vertical positions of the spindle mounting the milling cutter. In these machines the work is clamped on the table and is most frequently fed in the longitudinal direction.

*Universal milling machines* differ from the plain horizontal type in that their horizontal tables can be swivelled about a vertical axis to allow helical grooves to be milled. In up-to-date models the table can be swivelled up to  $45^\circ$  or  $50^\circ$  in either direction.

*Specialised milling machines* include planer-type milling machines with several spindles, face millers, circular continuous milling machines with rotary tables, continuous drum type machines with rotary drums and tracer-controlled machines for milling contoured surfaces.

*Single-purpose milling machines* include thread millers, keyway milling machines, unit-type and rack milling machines.

An external view of a universal milling machine is illustrated in Fig. 218a. Here the column 2 is secured on the base 1. At the top of the column is the overarm 3 on which the arbour support 4 is clamped. The flange-type drive motor 5 is mounted on the rear of the column; it powers spindle 6 through the speed gearbox arranged inside the column. Cutter 7 is clamped on arbour 8; it accomplishes the primary cutting motion (rotation about a horizontal axis). Knee 9 may travel vertically along ways on the front of the column. The knee carries the saddle 10 for cross travel, swivel-base saddle plate 11 and table 12.

Longitudinal, cross and vertical feeds may be imparted to the workpiece by means of the feed gearbox. All three types of feed may be accomplished either by power or by hand, in the latter case from the corresponding handwheels and crank handles. The feed gearbox is powered from the separate motor 13.

The gearing diagram of the model 6B82 universal milling machine is shown in Fig. 218b. The most expedient cutting speed in milling is selected by changing the gearing ratio  $i_{zg}$  of the speed gearbox by (1) shifting the sliding cluster gear 16-22-19 along shaft II and (2) shifting sliding cluster gears 26-37 and 82-19 and gear 47 along shaft IV. Thus 18 different speeds of spindle V are obtained in a range from 30 to 1500 rpm and transmitted to arbour 8 and cutter 7.

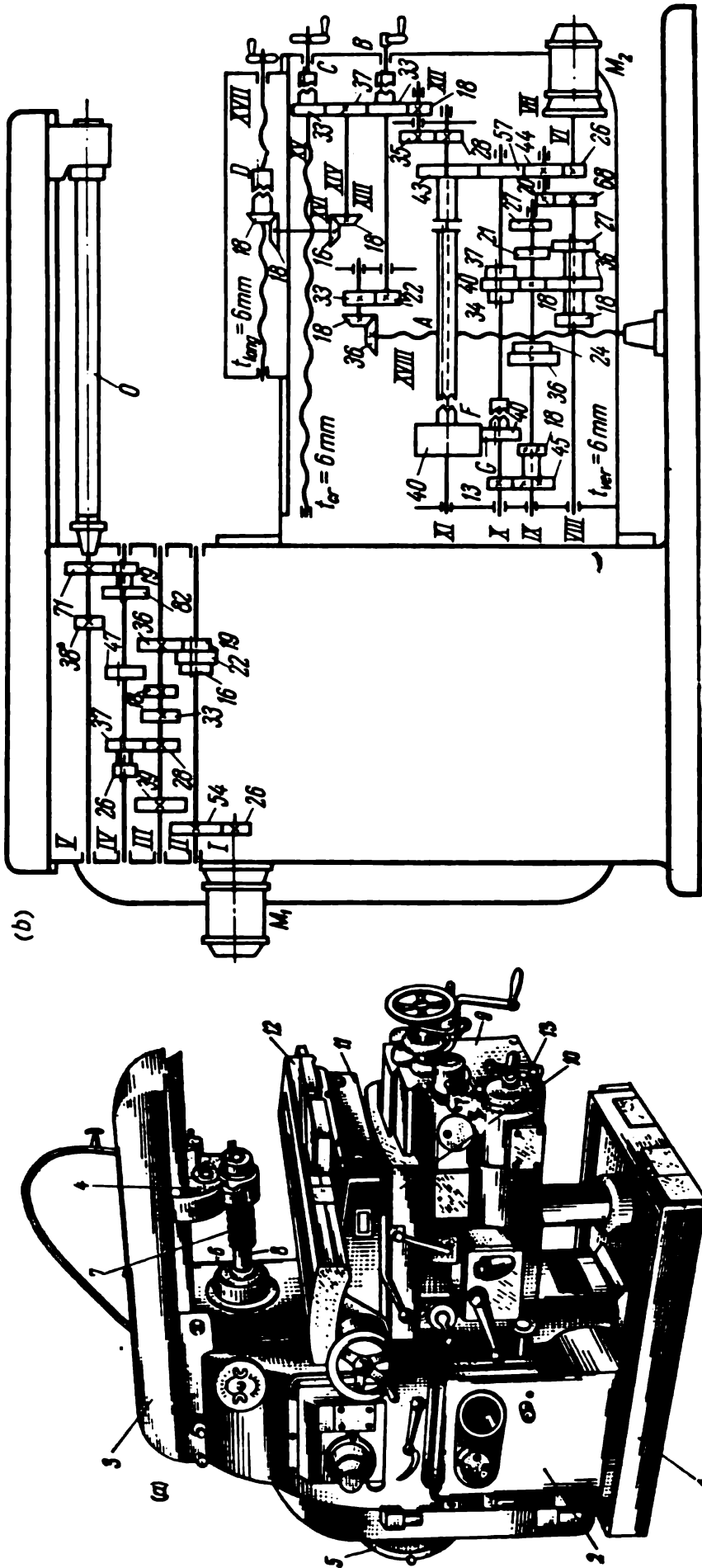


Fig. 218. Universal milling machine, model 6B82:  
(a) external view; (b) gearing diagram

If the speed  $n_m$  of the motor  $M_1$  is known, the equation of the whole spindle drive gear train can be expressed as

$$n_{sp} = n_m i_{sg} \text{ rpm,} \quad (120)$$

where

$i_{sg}$  = gearing ratio of the speed gearbox,  
 $n_{sp}$  = spindle speed, in rpm.

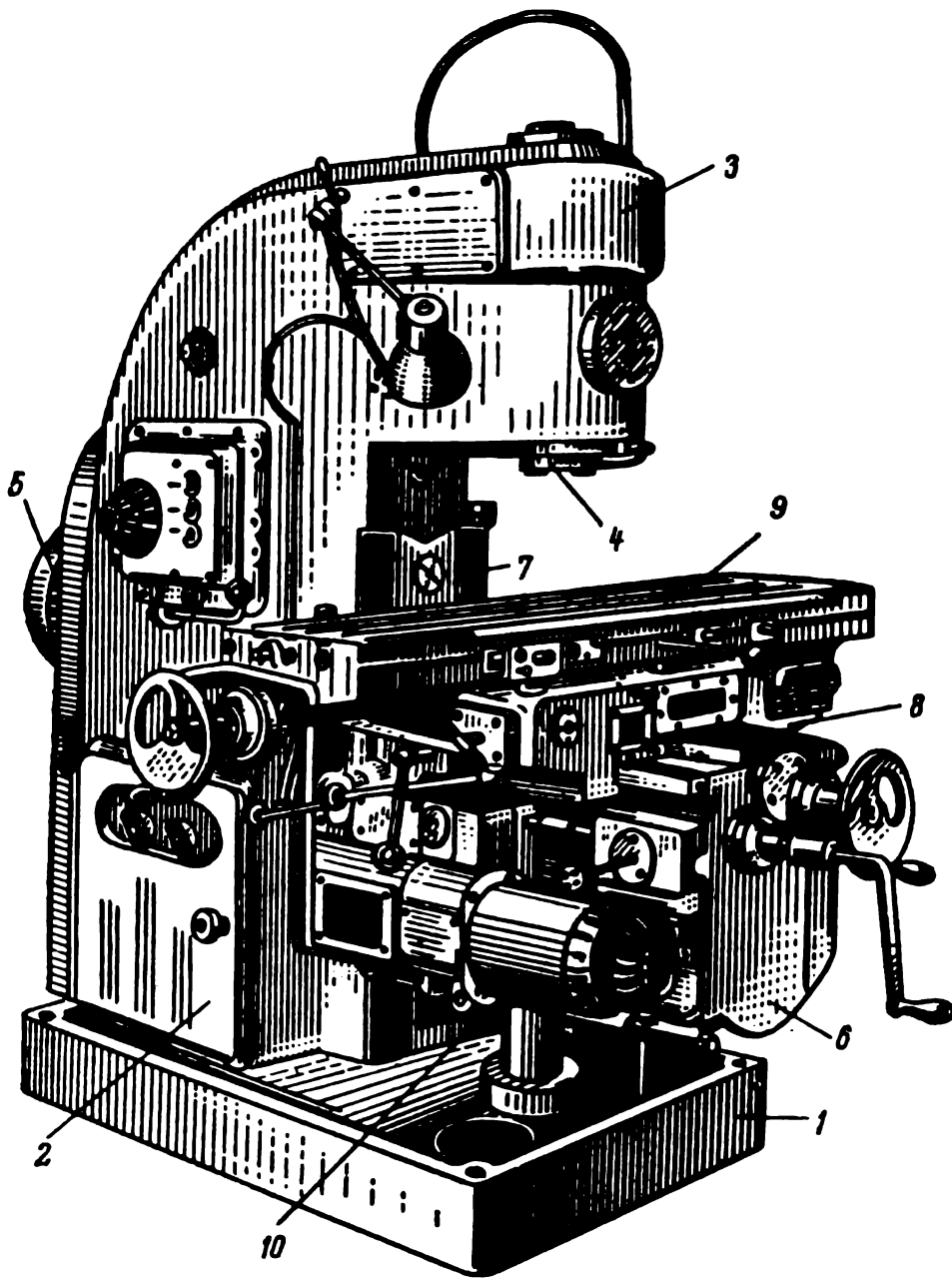


Fig. 219. Vertical-spindle milling machine, model 6H12

The feed mechanism is driven from motor  $M_2$ . The longitudinal, cross and vertical feeds are selected by shifting a number of sliding cluster gears and by engaging and disengaging clutches mounted on the shafts (V/ through X//) of the feed gearbox. By the corresponding engagements, 18 different feeds are obtained and transmitted as

required to the lead screws for longitudinal (XVII), cross (XV) or vertical (XVIII) feed.

Fig. 219 illustrates a vertical-spindle milling machine which uses face milling cutters, and mills and other shank-type cutters as tools.

The column 2 is bolted to base 1. The spindle head 3 with vertical spindle 4 is at the top of the column. The primary cutting motion (rotation about a vertical axis) is imparted to a face milling cutter or end mill clamped in the spindle or on the spindle nose. The spindle is powered from motor 5 through the speed gearbox and an additional pair of bevel gears in the head. The model 6H12 vertical milling machine has 18 different spindle speeds ranging from 30 to 1500 rpm.

Knee 6 travels up and down along ways 7 on the column. Saddle 8 travels in the crosswise direction along ways on the top of the knee and, in turn, has ways for longitudinal travel of the table 9.

Table travel in the longitudinal, cross and vertical directions is powered from the separate motor 10 through the feed gearbox located in the lower part of the knee. The feed gearbox of the given model provides 18 different table feeds; longitudinal and cross feeds from 23.5 to 1180 mm per min and vertical feeds from 8 to 320 mm per min. Both working feed and rapid traverse, obtained through separate gear trains, are available in all three directions.

### 34-4. Principal Types of Milling Operations

Milling machines can be employed to produce flat surfaces; slots, grooves and splines; all possible types of contoured surfaces; toothed gears, helical grooves and flutes, and others.

**Milling flat surfaces.** *Horizontal* flat surfaces may be milled on plain and vertical-spindle milling machines with plain and face milling cutters. Depending upon its size and shape the work is clamped on the table either in a special machine vise (Fig. 220a) or by means of T-bolts, strap clamps and pads (Fig. 220b). Work may also be held in various types of milling fixtures.

*Vertical* flat surfaces are milled on the same types of plain and vertical-spindle machines except that side milling cutters (Fig. 220c) or end mills and face milling cutters are used. Such surfaces may also be milled on planer-type machines with face milling cutters (Fig. 220d). The work is clamped on the table in the same manner as for horizontal flat surfaces.

*Inclined* flat surfaces are milled on plain milling machines with single angle cutters (Fig. 220e) or on vertical-spindle models with the spindle swivelled to the required angle  $\alpha$ , using a face milling cutter (Fig. 220f). This spindle adjustment is possible only on swi-

vel-head machines having a column-mounted head that can be swivelled in a vertical plane parallel to the column face.

*Recesses* are milled on vertical-spindle machines with end mills (Fig. 220g); in this case the two perpendicular surfaces are machined simultaneously.

**Milling slots, grooves and splines.** Rectangular, T- and dovetail slots are milled, as a rule, on vertical-spindle machines with the ap-

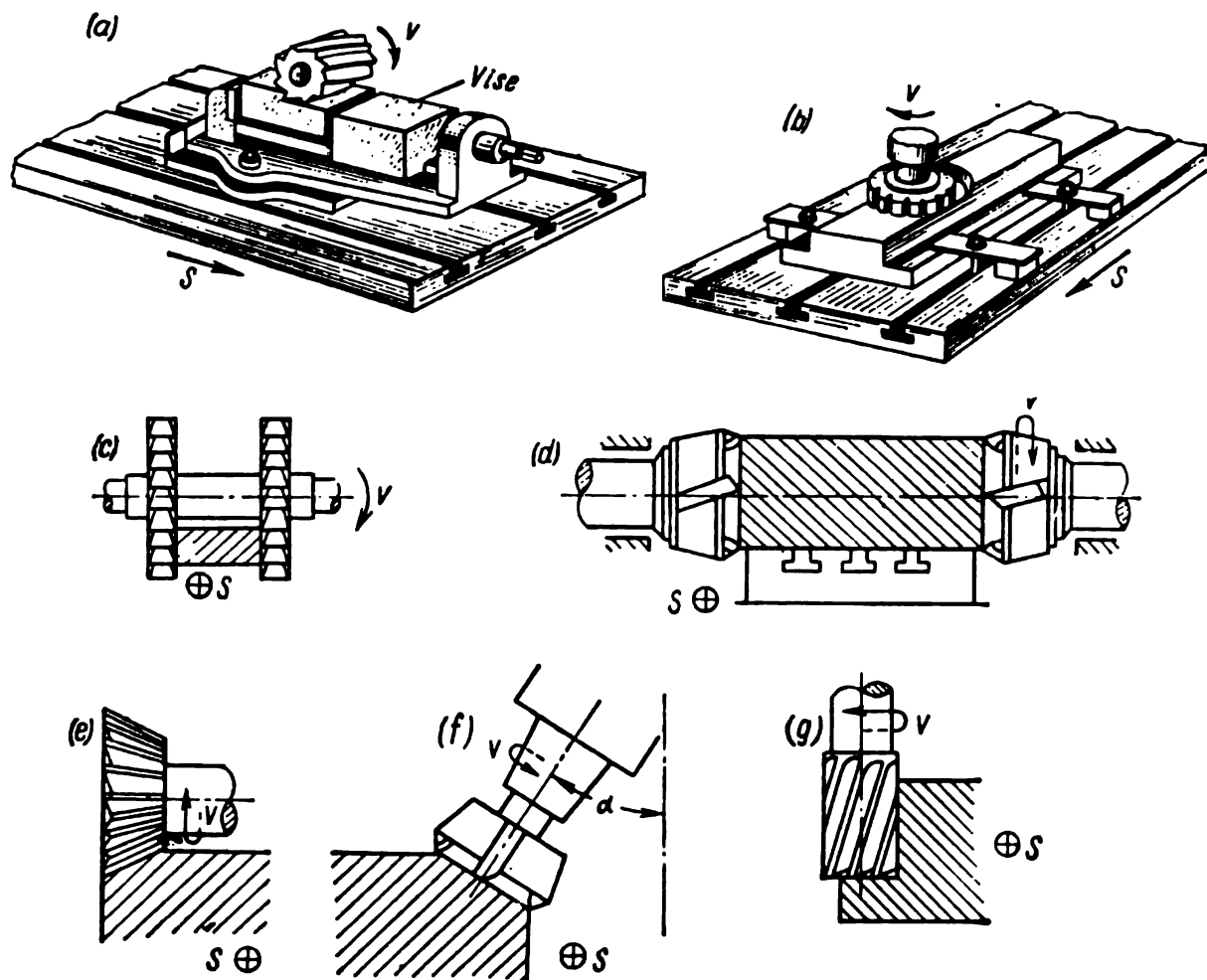


Fig. 220. Milling flat surfaces

propriate shank-type milling cutters (Fig. 221a and b). Rectangular slots can also be produced on plain milling machines with side milling cutters.

End mills and special cutters are used to mill *keyways* of rectangular cross section. This operation is performed on vertical-spindle machines or on keyway milling machines. Sometimes keyways are milled on plain milling machines with side milling cutters.

In keyway milling, workpieces of the shaft type are clamped in special vises with locating vee-blocks and swinging jaws.

*Splines* may be milled on plain milling machines by means of single- and double-angle cutters.

**Milling contoured surfaces.** Formed milling cutters with a profile coinciding with that of the workpiece are used to mill contoured surfaces of relatively small width on plain milling machines (Fig. 221c). Surfaces of more complex shape are milled by a gang of simple cutters (Fig. 221d) or to a template on a tracer-controlled machine.

Three-dimensional contoured surfaces (on dies, etc.) are machined on tracer-controlled contouring machines.

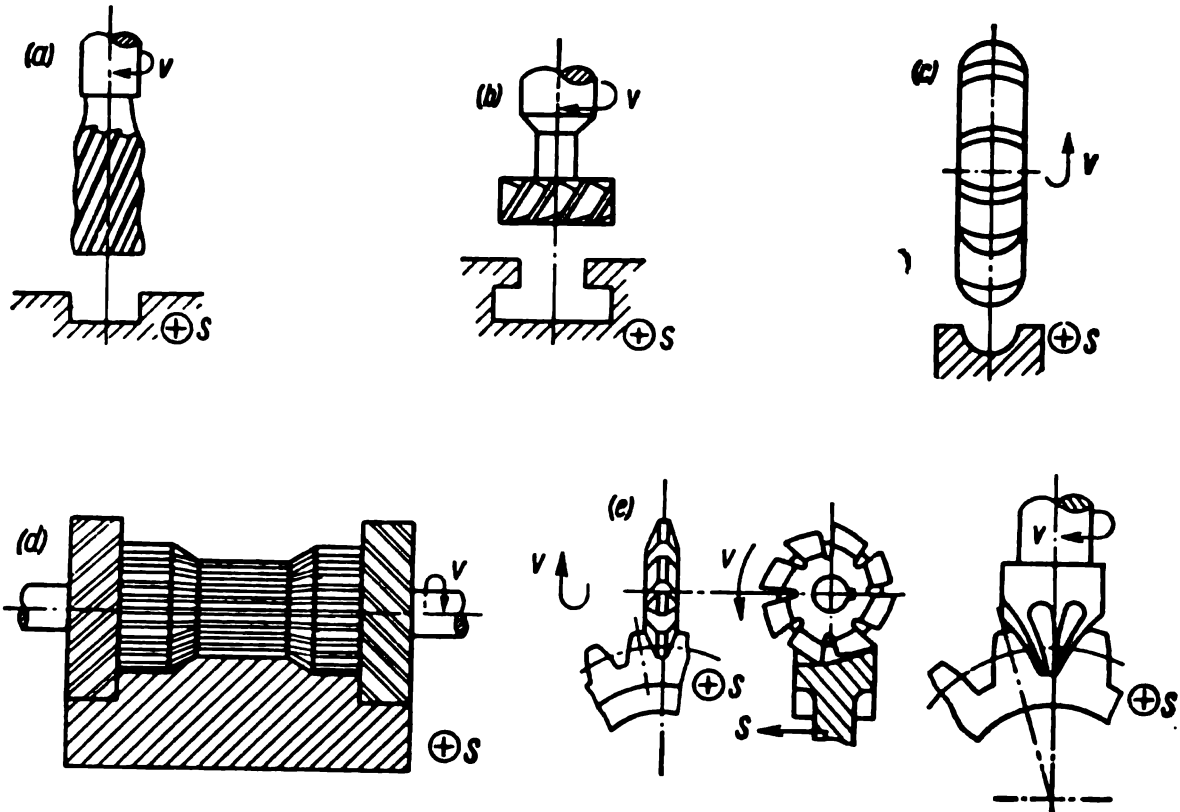


Fig. 221. Milling slots, grooves and contoured surfaces

**Cutting toothed gears and helical grooves.** Spur and helical gears are cut by form-relieved gear milling cutters on plain milling machines or, in some cases, by end-mill type gear milling cutters on vertical-spindle machines (Fig. 221e). The cutter profile must exactly coincide with the tooth space of the gear. Hobbing machines, gear shapers and other machine tools are used to cut gears for lot and mass production.

Special attachments to milling machines called dividing heads are used to index the gear blank through the required angle from one tooth space to the next, i. e., they divide a circle into a definite number of equal parts.

There are universal (with simple and differential indexing), plain, optical and other types of dividing heads. The first are more widely employed; in addition to indexing the work they can be used to rotate it for milling helical grooves or the tooth spaces of helical gears.

A universal dividing head (Fig. 222a) consists of the body 1, swivel block 2, work spindle 3 with centre 4, index plate 5 and index crank 6 with latch pin 7.

In simple indexing the work is turned through the required angle by turning crank 6 with latch pin 7 in relation to the stationary index plate 5. The index plate is mounted on shaft 8 passing through sleeve 9 (Fig. 222b). Dividing heads are usually furnished with a set of three index plates, each having six circles of equally spaced holes. The first plate has 15, 16, 17, 18, 19 and 20 hole circles, the second

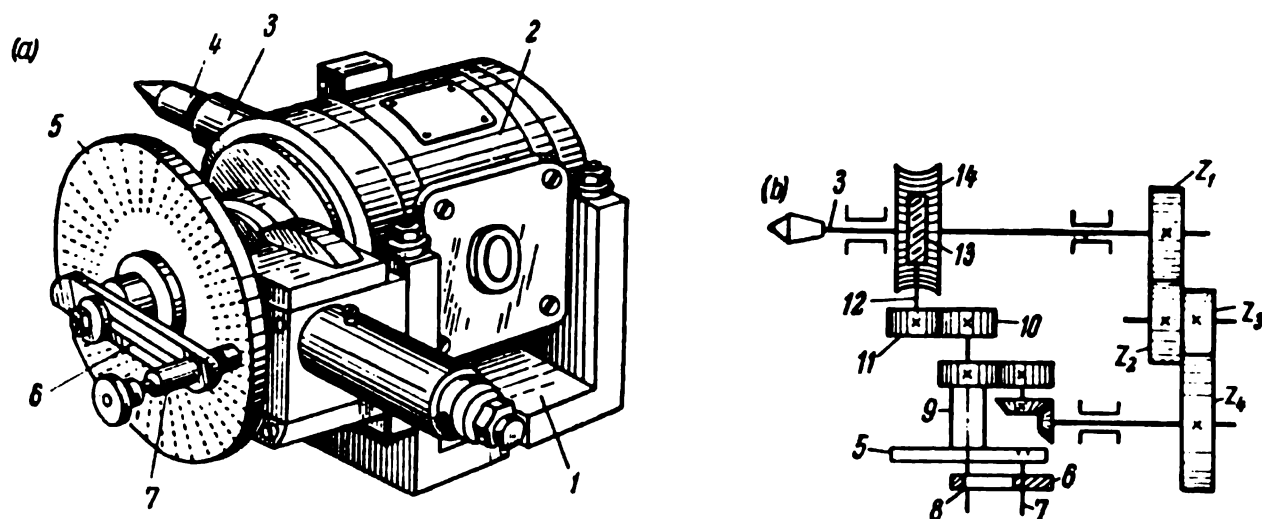


Fig. 222. A universal dividing head:  
(a) external view; (b) gearing diagram

plate has 21, 23, 27, 29, 31 and 33 hole circles and the third—37, 39, 41, 43, 47 and 49 hole circles. Each circle of holes divides the circumference of the plate into equal parts.

Gear 10 is mounted on the other end of shaft 8. It meshes with gear 11 mounted on worm shaft 12. Worm 13 meshes with worm wheel 14 which is rigidly mounted on spindle 3 of the head.

Worm wheel 14 has 40, 60, 80 or more rarely 120 teeth; most frequently it has 40 teeth. Then the ratio of the worm gearing is  $i = \frac{1}{40}$ . This means that one complete revolution of worm gear 14 and spindle 3 will require 40 turns of the crank, one half revolution will require 20 turns, one fourth revolution—10 turns, etc.

If the number of teeth (or rather spaces) through which the gear blank must be indexed is denoted by  $z$  and the number of teeth on the worm wheel by  $k$ , then the number of turns that the crank must be rotated for each indexing can be found from the formula

$$n = \frac{k}{z}. \quad (121)$$

When none of the index plates has a hole circle which would enable the work to be divided by the simple indexing method, a more in-

volved method called differential indexing is employed. For this purpose the index plate is linked to the spindle through the change gears  $z_1$ ,  $z_2$ ,  $z_3$  and  $z_4$  of a quadrant (Fig. 222b).

Helical grooves are milled on a universal milling machine. In this case the table is swivelled to the helix angle and the work is rotated during milling by the dividing head which is linked by gearing to the lead screw of the table.

## Chapter 35

### PLANING, SHAPING AND SLOTTING PRACTICE

#### 35-1. The Planing Process

*Planing, shaping and slotting* are processes for machining horizontal, vertical and inclined flat and contoured surfaces, slots, grooves and other recesses by means of special single-point tools. The difference between the three processes is that in planing the work is reciprocated and the tool is fed into and across the work while in shaping and slotting the tool is reciprocated and the work is fed into the cutting tool. Tool travel is horizontal in shaping and vertical in slotting.

Cutting is intermittent in all three processes since the tool (or work in planing) makes alternating working and idle-return strokes.

The cutting speed  $v_w$  in planing, shaping and slotting is the speed of the tool (or the work) in the direction of the primary cutting motion during the working stroke when metal is being cut.

The cutting speed may be either constant or variable depending upon the design of the machine tool.

Shapers with crank and slotted arm mechanisms have a variable working speed  $v_w$  as well as a variable return speed  $v_r$  of the tool. The speed has its minimum value at the beginning and end of the stroke and reaches its maximum value in the middle of the stroke. The cutting speed for these machines can be determined from the formula

$$v_w = \frac{Ln(1+m)}{1000} \text{ m per min,} \quad (122)$$

where  $L$  = length of the ram stroke, in mm,

$n$  = number of full strokes per min (a full stroke comprises the working and return strokes),

$m = \frac{v_w}{v_r}$  = ratio of the working stroke speed to the return stroke speed;  
for short ram strokes  $m=0.75$ , on an average.



Planers with a rack and pinion drive for the table have a constant cutting speed which can be determined from the formula

$$v_w = \frac{2 Ln}{1000} \text{ m per min.} \quad (123)$$

The permissible cutting speed in planing and allied processes depends upon the properties of the material being machined, tool material and geometry, rate of feed, depth of cut, and the length of the table stroke with the work or the ram stroke with the tool.

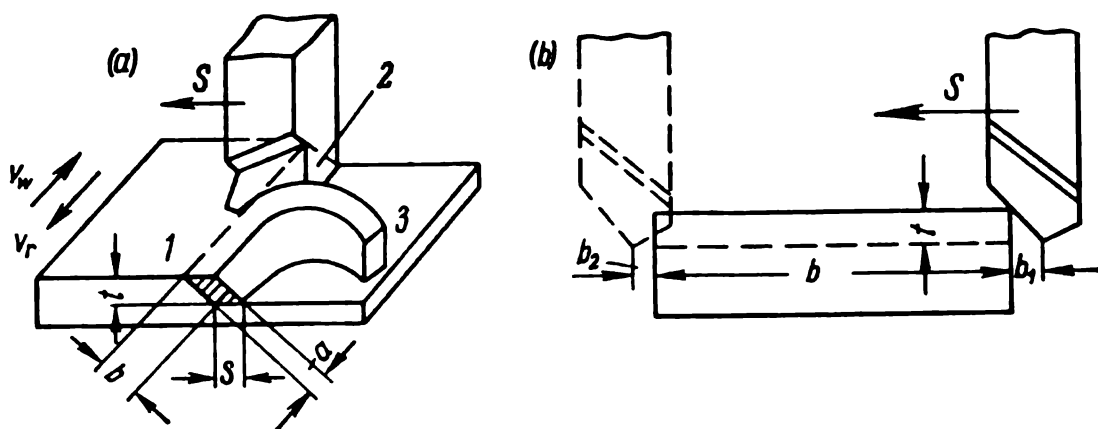


Fig. 223. Schematic diagram of the planing process and length of tool travel in the direction of the feed:

1—work surface; 2—cutting surface; 3—machined surface

The feed  $s$  in planing, shaping or slotting (Fig. 223a) is the relative movement of the tool or work in a direction perpendicular to the primary cutting motion per full stroke and is expressed in mm per stroke. The feed movement always takes place at the end of the return stroke when the tool is not cutting the chip.

The depth of cut  $t$  (see Fig. 223a) is the thickness of the layer of metal removed in one cut or pass; it is measured in a direction perpendicular to the machined surface.

The cross-sectional area  $f$  of the undeformed chip (see Fig. 223a) is the product of the thickness  $a$  and the width  $b$  of the undeformed chip:

$$f = ab = st \text{ sq mm.} \quad (124)$$

Machining time in these processes is calculated on the basis of the length of tool or work travel in the direction of the feed (width of the work plus tool approach and overtravel) (Fig. 223b), number of full strokes per minute, rate of feed and the number of cuts.

### 35-2. Planing and Shaping Tools

The same single-point tools (Fig. 224a) are used in planing and shaping; they are similar to lathe tools and consist of the shank *A* and point *B*. The point is bounded by the face 1, side flank 2 and end flank 3 and has a side-cutting edge 4, end-cutting edge 5 and a nose 6.

The chief plan *N-N*, intersecting the side-cutting edge and perpendicular to the cutting surface (Fig. 224b), contains the side-rake angle  $\gamma$ , side-relief angle  $\alpha$ , lip angle  $\beta$  and cutting angle  $\delta$ . The plan angles include the plan approach angle  $\varphi$ , end-cutting-edge angle  $\varphi_1$  and the nose angle  $\epsilon$ .

The side-rake angle  $\gamma$  of planing and shaping tools may vary from  $5^\circ$  to  $20^\circ$  depending upon the hardness and strength of the material being machined.

The side-relief angle  $\alpha$  usually varies from  $6^\circ$  to  $14^\circ$ .

The plan approach angle  $\varphi$  for straight-planing tools is taken from  $30^\circ$  to  $75^\circ$ ; angle  $\varphi_1$ —from  $10^\circ$  to  $30^\circ$ . Angle  $\varphi_1$  equals  $2^\circ$  or  $3^\circ$  on slotting tools.

Planing tools are classified as right- or left-hand depending upon the location of the side-cutting edge; straight- or bent-shank according to the relative arrangement of the point and shank; straight-planing, facing, slotting and form tools according to their purpose (Fig. 225), and roughing or finishing according to the type of operation they perform.

Planing and shaping tools are made of high-speed steel or tipped with cemented carbides. In the latter case the shanks are made of carbon tool steel.

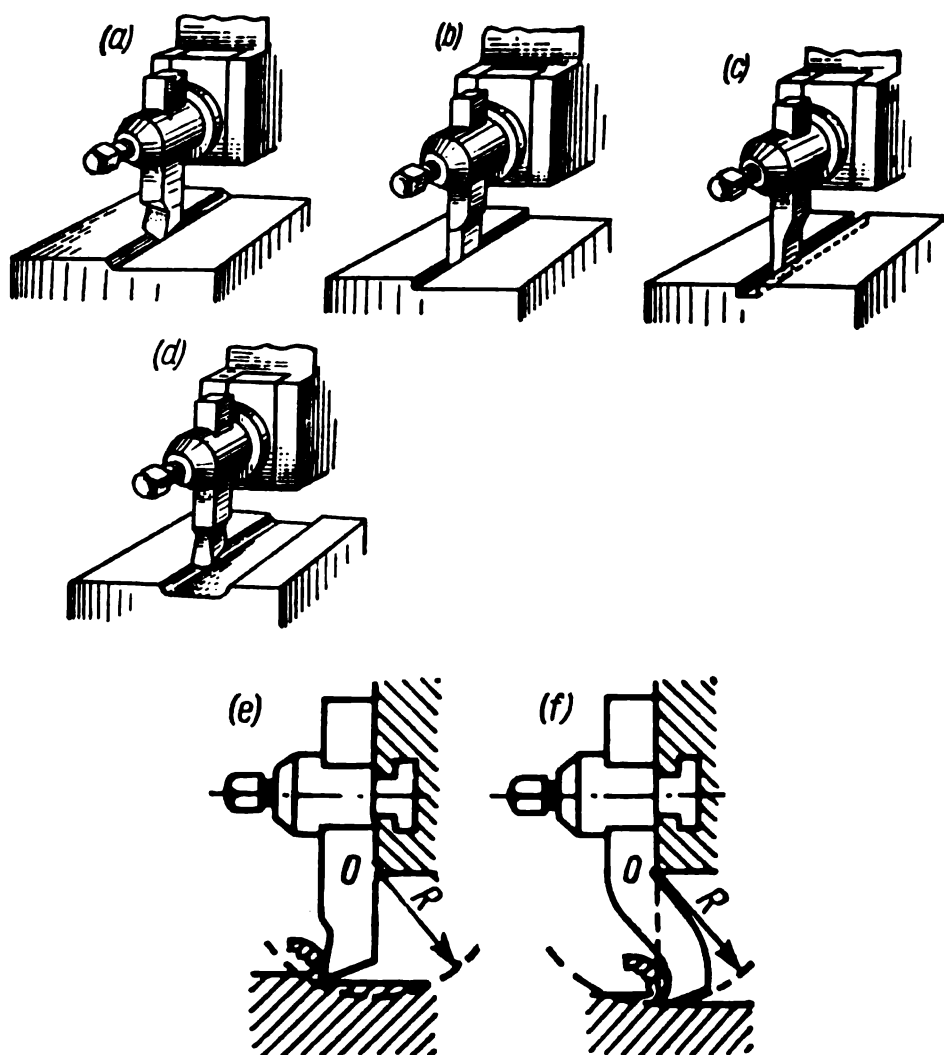
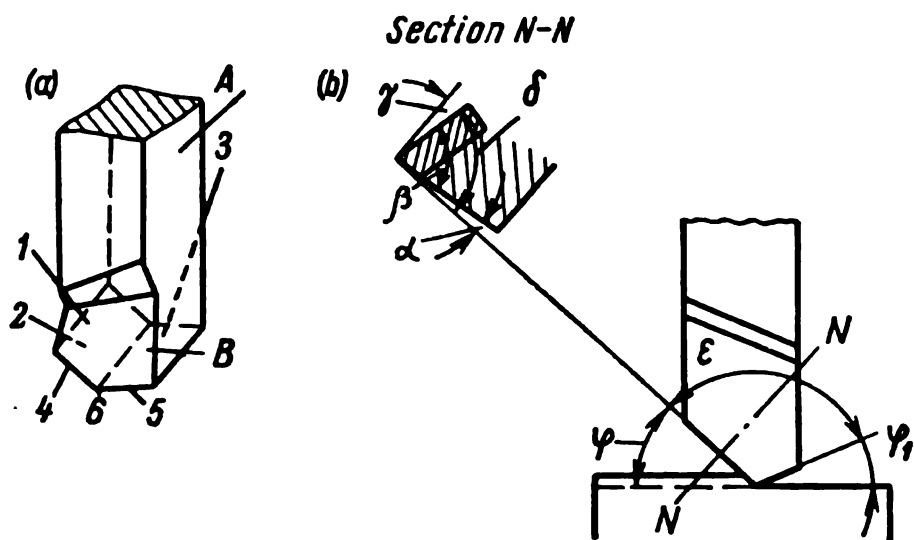
### 35-3. Shapers and Planers

Machine tools operating on the planing principle are classified according to their processing features as shapers, planers, slotters (vertical shapers) and specialised types.

Each such class includes several types distinguished by certain design features.

The shaper shown in Fig. 226 is designed for machining flat and formed surfaces on small-sized workpieces.

The column 1 of the shaper houses the speed gearbox and the crank and slotted arm mechanism which transmit power from motor 2 to ram 3 and table 4. Ram travel is the primary reciprocating motion while the intermittent cross travel of the table is the feed motion. The tool head 5, carrying the clapper box and tool holder 6, is mounted at the front end of the ram.





The slot with the adjustable clamp 7 serves to position the ram in setting up the shaper. The tool head has a toolslide and feed screw rotated by ball-crank handle 8 for raising and lowering the tool to adjust the depth of cut. A swivel motion of the tool head enables it to take angular cuts to machine inclined surfaces.

The workpiece is clamped either directly on the table or is held in a machine vise. By means of ratchet and pawl mechanism 9, driven from the crank and slotted arm mechanism, the table is fed cross-wise in a horizontal plane. The table is raised or lowered by elevating screw 10 which is operated by a crank handle located on the opposite side of the shaper. Support bracket 11 is provided to clamp the table rigidly during operation.

The required number of strokes per minute of the ram are set by shifting levers 12 to the corresponding positions.

Fig. 226*b* shows a gearing diagram of a crank shaper. Rotation of the motor is transmitted by a belt drive to speed gearbox *A* which provides six speeds to the bull gear  $z=102$  by various engagements of the sliding cluster gears mounted on shafts *I* and *III*. The crank and slotted arm mechanism consisting of the bull gear  $z=102$  and slotted arm *B*, linked to ram 3, converts rotary motion of the bull gear into reciprocating motion of the ram with the cutting tool. The length of the ram stroke depends upon the radius of the crankpin on the bull gear; the larger the crank radius (radius of pin 13 with the sliding block) the longer the ram stroke will be, and vice versa. This adjustment also alters the speed of the ram.

To select the most expedient cutting speed the number of full strokes per minute  $n_{fs}$  is found from the formula

$$n_{fs} = n_m i_{sg}, \quad (125)$$

where  $n_m$  = motor speed, in rpm,

$i_{sg}$  = gearing ratio of the speed gearbox.

The required gearing ratio and, consequently, the required number of ram strokes are obtained by shifting the sliding cluster gears in the speed gearbox to the corresponding engagements with levers 12.

Table feed is obtained by transmission of rotation from the bull gear  $z=102$  through the pair of gears 36-36, connection 14, pawl carrier lever 15, pawl 16 and ratchet wheel 17 to the cross feed screw.

The toolslide with the tool is fed vertically either manually or by power.

A double-housing planer is illustrated in Fig. 227. This planer has a bed 1 at the sides of which two vertical housings 2 and 3 are arranged. The housings are connected together at the top by cross member 4. Table 5 travels along the ways of the bed and has T-slots in its upper surface for clamping the workpiece. The travel of the table with the work is the primary reciprocating motion (cutting mo-

tion) of the planer. The table is powered from a variable-speed direct-current motor through a reducing gear and a rack-and-pinion drive.

Crossrail 6 can be traversed vertically along ways on the housings by means of a drive powered from a separate motor. The crossrail carries tool heads 7 and 8 which mount the tools for planing horizontal surfaces. Both tool heads have cross feed along the crossrail ways

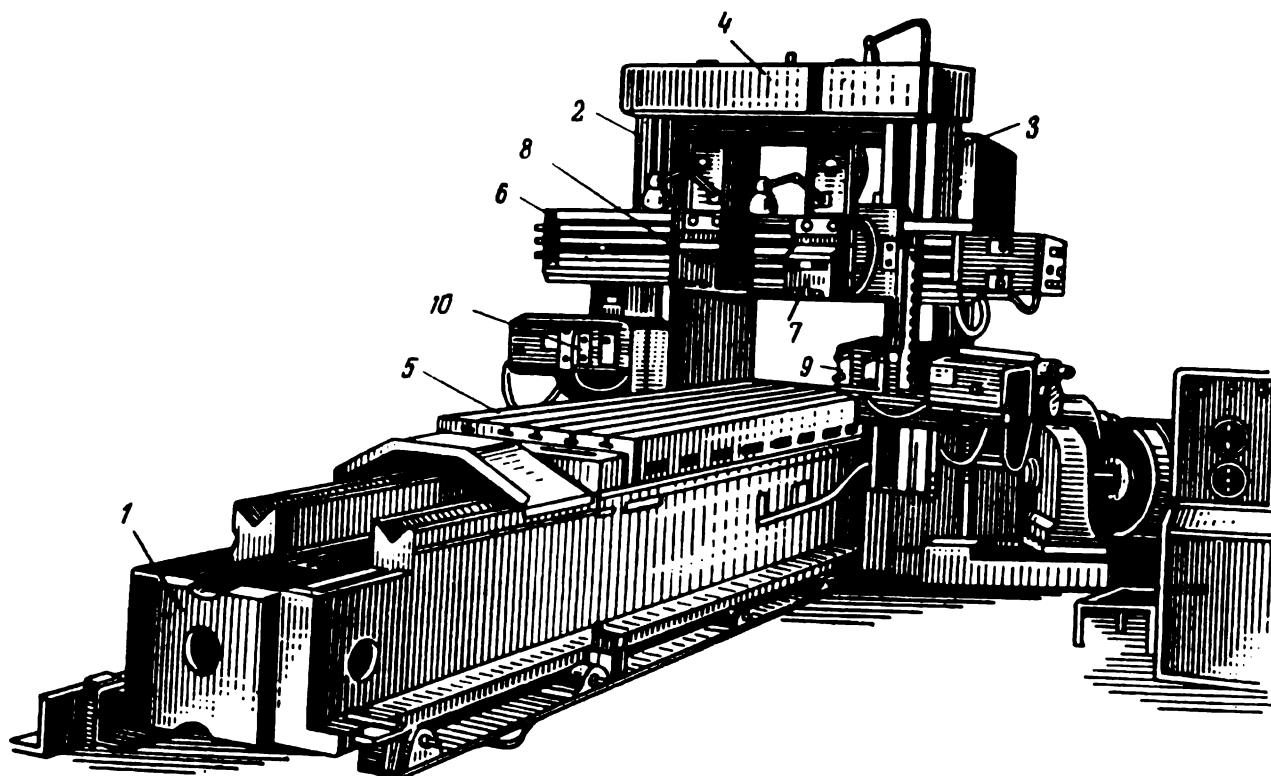


Fig. 227. Double-housing planer, model 7231A

powered by their motors. The toolslides with the tool posts on the tool heads can be traversed vertically to feed the tools down for a new cut.

Vertical surfaces of the workpiece are planed by tools clamped in the side tool heads 9 and 10 which travel vertically along housings 2 and 3. The four tool heads of the planer operate independently of each other.

The Soviet planer, model 7231A is employed in planing large workpieces weighing up to 5 tons. The maximum length and width planed are 3000 mm and 1000 mm, respectively.

#### 35-4. Principal Types of Planing and Shaping Operations

*Horizontal flat* surfaces are machined on shapers or planers with straight-planing tools. The maximum possible feed and depth of cut are used in roughing operations (Fig. 228a). Moreover, special

tool holders clamping two, four and more tools are employed in planing heavy workpieces to more completely utilise the available power of the machine tool and to increase the productivity. Finishing operations are done with broad-nose tools, as a rule; the depth of cut  $t$  should be within 0.5 to 1 mm.

*Vertical flat* surfaces are machined on planers with straight-planing tools clamped in the side tool heads on the housings. On shapers

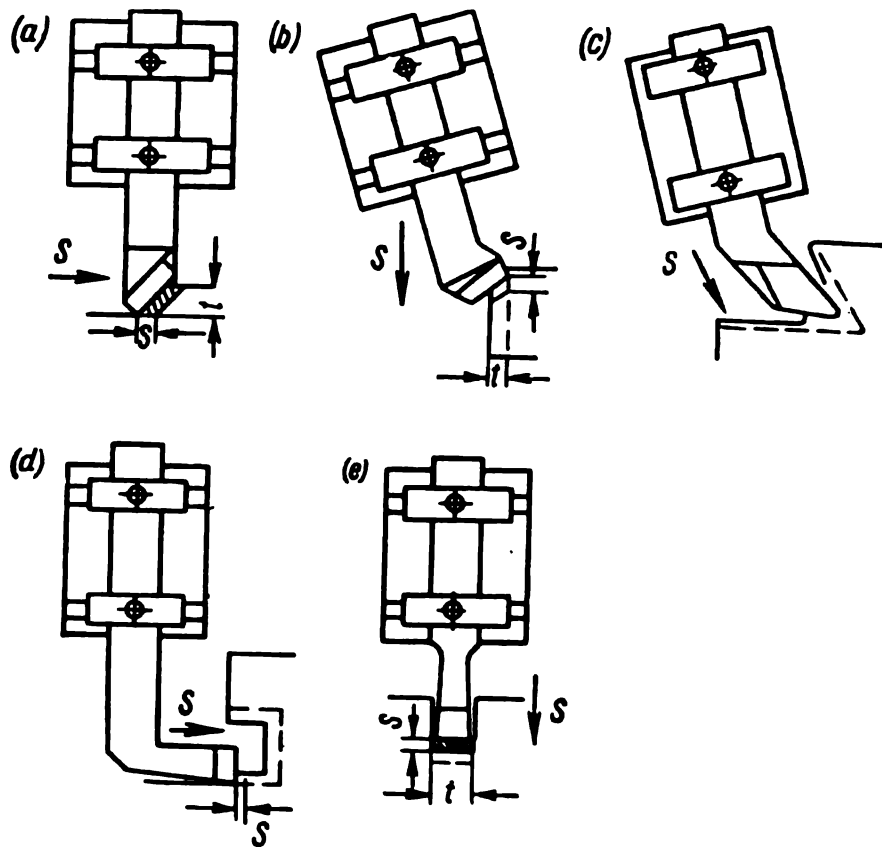


Fig. 228. Main types of flat surfaces, slots, recesses, and grooves machined by planing or shaping

such surfaces are machined with straight-planing or facing tools set at a certain angle to the surface to be machined (Fig. 228b).

*Inclined flat* surfaces (Fig. 228c) are machined on planers and shapers with straight-planing and facing tools of special shape and/or an angular position of the tool slide. On a shaper the tool is fed manually parallel to the work surface.

*Slots and grooves* (Fig. 228d and e) are most conveniently machined by means of slotting and undercutting tools using either vertical or horizontal feed.

## Chapter 36

### GRINDING PRACTICE

#### 36-1. The Grinding Process

In the majority of cases grinding is considered to be a finishing operation employed to provide high dimensional accuracy and a fine surface finish.

The most important types of grinding are cylindrical, internal, surface, external centreless, and others.

*The cutting speed  $v_{wh}$*  in cylindrical grinding is the peripheral speed of the grinding wheel. It is expressed in metres per second (m/sec) and is selected in the range from 30 to 50 m per sec and frequently even higher. In this type of grinding the workpiece rotates as well and its peripheral speed may vary from 15 to 50 metres per minute.

*The feed  $s$*  in cylindrical grinding is the longitudinal movement of the workpiece per revolution; it is expressed as millimetres per revolution of the workpiece (mm/rev).

*The depth of cut  $t$*  (cross feed) is the thickness of the layer of metal removed in one pass (Fig. 229a).

*Machining time  $T_m$*  in cylindrical grinding is the time during which grinding is done and its calculation is based on the length of longitudinal table travel, grinding allowance on the workpiece, depth of cut in each pass, workpiece speed, longitudinal feed and the specified accuracy.

*Cylindrical grinding* between centres may be performed by the traverse (longitudinal feed) and plunge-cut (cross feed) methods as well as by the so-called full-depth method.

In traverse grinding (Fig. 229a) wheel rotation is the primary rotary cutting motion  $v_{wh}$ . In addition, there is work rotation  $v_{wk}$  and longitudinal linear feed  $s_{lr}$  of the work past the wheel (or the wheel past the work). The depth of cut  $t$  is obtained by cross feed of the wheel into the work, or vice versa.

The primary rotary motion  $v_{wh}$  is wheel rotation in plunge-cut grinding as well but here the wheel is fed into the work with the cross feed  $s_{p,r}$ . The workpiece only rotates at the speed  $v_{wk}$ . This method may be applied when the length of the surface to be ground is shorter than the wheel width.

The full-depth method involves the setting of the grinding wheel to almost the full grinding allowance which is removed in one pass with longitudinal feed of the work  $s_{lr}$ . Wheel rotation  $v_{wh}$  and work rotation  $v_{wk}$  are the same as in the preceding methods. The method



is usually applied to relatively short surfaces of rigid shaft-type workpieces.

*Internal grinding* (Fig. 229b) is employed for machining holes in comparatively short workpieces held in a chuck or special fixture. The movements involved in this type of grinding are wheel rotation, workpiece rotation, and wheel feed.

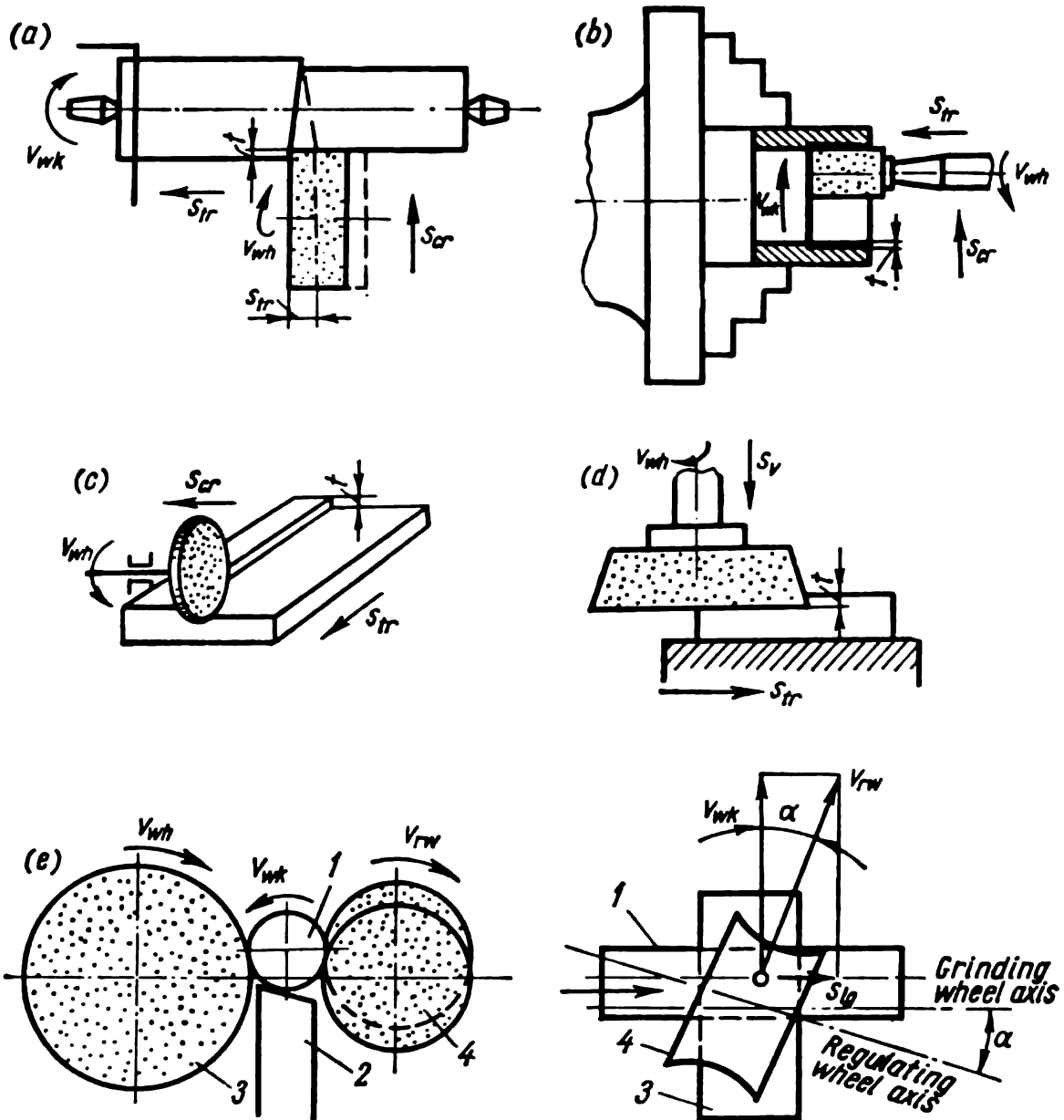


Fig. 229. Principal types of grinding

the primary rotary motion  $v_{wh}$ , wheel feed  $s_{fr}$  in a longitudinal direction and rotation  $v_{wk}$  of the workpiece. The depth of cut  $t$  is obtained by cross feed of the wheel.

In *surface grinding* the work may be ground by either the periphery or by the end face of the grinding wheel.

The primary rotary motion  $v_{wh}$  in grinding with the periphery of the wheel (Fig. 229c) is wheel rotation, as in all other grinding pro-

cedures; the wheel also has cross feed  $s_{cr}$  while the workpiece, clamped on the machine table, reciprocates with the longitudinal feed motion  $s_{lg}$ . The depth of cut  $t$  is set by vertical feed of the wheel. In other types of surface grinding machines the table with the workpiece rotates in a horizontal plane (as in a vertical turning and boring mill) and the wheel is fed radially in reference to the wheel.

The movements of the work in grinding with the end face of the wheel (Fig. 229d) are the same as when the wheel periphery is used; the primary rotary motion  $v_{wh}$  is wheel rotation about the vertical axis. The depth of cut is provided by vertical feed of the wheel along its axis.

*External centreless grinding* (Fig. 229e) is done by passing the cylindrical workpiece 1, supported by the work rest blade 2, with longitudinal feed between the grinding wheel 3 and regulating wheel 4. The grinding wheel does the actual grinding while the regulating wheel, or feed wheel, is inclined at an angle  $\alpha$  in reference to the grinding wheel axis and its functions are to control work rotation and to provide longitudinal feed of the work past the grinding wheel. Due to the inclination of the regulating wheel its peripheral speed  $v_{rw}$  is resolved into two components—work rotation speed  $v_{wk} = v_{rw} \cos \alpha$  and the rate of longitudinal feed  $s_{lg} = v_{rw} \mu \sin \alpha$ , where  $\mu$  is a coefficient which accounts for slipping of the work in reference to the regulating wheel ( $\mu = 0.94$  to  $0.98$ ).

Angle  $\alpha$  is usually taken from  $1^\circ$  to  $5^\circ$ ; the larger angle  $\alpha$ , the higher the rate of feed will be, and vice versa.

The periphery of the regulating wheel is not cylindrical but slightly concave (actually a hyperboloid of revolution) to improve contact between the wheel and the work.

If the regulating wheel axis is set parallel to that of the grinding wheel, then  $s_{lg} = 0$  and there will be no longitudinal feed of the work. This feature is used in the centreless grinding of workpieces with shoulders or heads by the infeed method.

Each of the types of grinding discussed above involves the use of a grinding wheel of specific shape and is performed on a grinding machine of appropriate design.

## 36-2. Grinding Wheels

Grinding wheels of all shapes are composed of carefully sized abrasive grains held together by a bonding material. The actual cutting process in grinding is the result of the action of the abrasive grains. Pores between the grains and bond allow the grains to act as separate single-point tools and provide chip clearance to prevent clogging of the wheel.

Grinding wheels are distinguished by their shape and size, type of abrasive material, grain size, bond, grade (hardness) and structure.

According to their shape wheels are classified as: (1) straight (Fig. 230a) for cylindrical, internal, centreless and surface grinding, (2) bevelled-face straight (Fig. 230 b and c) for grinding thread, gear

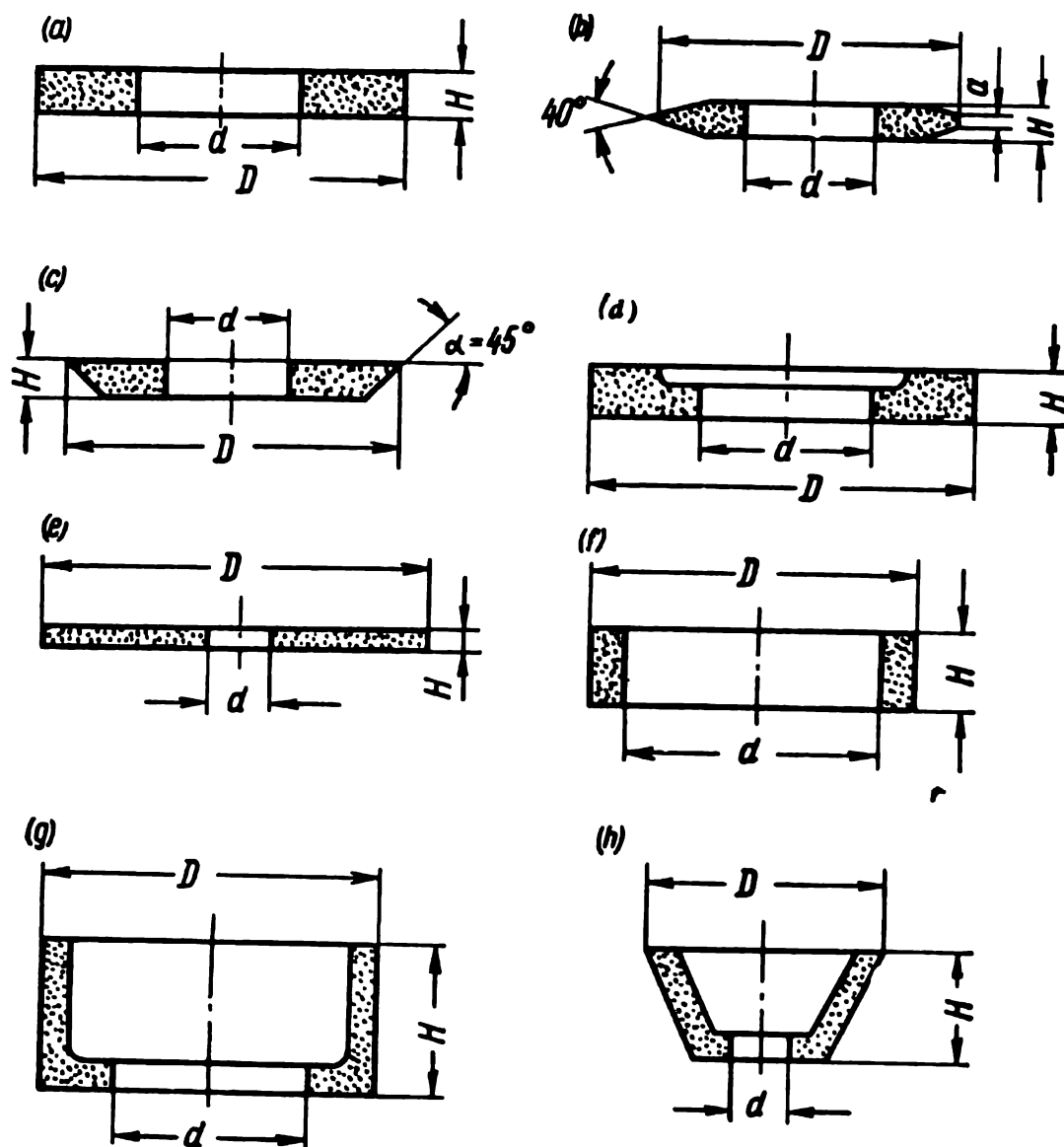


Fig. 230. Grinding wheel shapes

teeth, etc., (3) straight recessed (Fig. 230d) for cylindrical grinding and facing, (4) abrasive-cutting types from 0.5 to 5 mm thick (Fig. 230e) for abrasive-cutting-off and slotting operations and (5) cylinder, straight cup and flaring cup (Fig. 230 f, g and h) for surface grinding with the end face of the wheel.

The principal dimensions of a grinding wheel are the outside diameter  $D$ , bore  $d$  and the width  $H$ . Straight wheels are available with an outside diameter from 3 to 1100 mm and with width from 6 to 200 mm.

Grinding wheels may be made of natural abrasives—corundum and emery—or of artificial substances, such as aluminium oxide (artificial corundum), silicon carbide (sometimes called carborundum) and boron carbide.

The grains of abrasive materials possess very high hardness and heat resistance, have sharp edges produced when the lumps are crushed and are able to cut extremely hard materials (chilled skin of castings, hardened steel, etc.).

The surface quality of a ground workpiece depends largely on the grain size of the wheel; coarse and medium-sizes are normally used for roughing and semifinishing operations and fine-sizes for finishing.

Grain size is denoted by a number indicating the number of meshes per linear inch of the screen through which the grains pass when they are graded after crushing.

Grinding wheels are made of grains in sizes from 10 to 90, abrasive powders in sizes from 100 to 320 and abrasive flour in sizes from M28 to M5 (in the last case the number after the letter M is the grain size in microns (1 micron=0.001 mm). Coarse-grained wheels (up to 24) are used in rough grinding, medium-grained wheels (up to 60)—for ordinary grinding and tool grinding, and fine-grained wheels (up to 120)—for finishing grinding. Wheels with a very fine grain (120 to 320 and abrasive flours) are employed to grind threads.

The bond of a grinding wheel, i. e., the binding material that holds the abrasive grains, may be either of an organic or nonorganic type. The bonds are vitrified, silicate, oxychloride, resinoid and rubber.

The grade, or hardness, of a grinding wheel has no reference to the hardness of the abrasive grains—it is simply the resistance of the bond to the external forces striving to tear the grains from the wheel. Soviet standards stipulate seven grades of grinding wheels: soft—M, medium soft—CM, medium—C, medium-hard—CT, hard—T, very hard—BT and extremely hard—4T.

Selecting the right wheel for a definite job is of prime importance for the grinding process and particularly in respect to the self-dressing properties of the wheel. Self-dressing is breaking away of the worn dull grains to expose new sharp grains. If an excessively hard wheel is selected for a given job or material the dull grains will not be pulled from the bond quickly enough and no self-dressing will occur. This leads to the clogging of the wheel and burns on the ground surface. Too soft a grade will result in grains being pulled from the bond before they are dulled and the wheel loses its shape prematurely. As a rule, hard grades are used to grind soft materials, and vice versa.

The cutting properties of a clogged or loaded wheel are restored by dressing with star-type dressers, cemented carbide rollers, abrasive wheel dressers or diamond tools. Truing, another operation done

with the same devices, is shaping or cutting away the wheel to make its face concentric, its sides flat and parallel, or to alter its shape for grinding special contours.

Structure of a grinding wheel is characterised by the quantitative relation and arrangement of the grains, bond and voids (pores) in a wheel (or, more simply, the spacing of the grains). Structure is designated by numbers from 0 to 12. In a structure higher by one number the volume of the grains is reduced by 2 per cent, the volume of the binding material is increased by 2 per cent while the total volume of the voids remains constant through their size is altered. The lower numbers indicate closer spacing or denser structures.

Fine-pored wheels (dense structures 4 and 5) are applied in grinding hard and brittle materials when a high quality of surface finish is specified; wheels with medium-sized pores (structure 6) are used for cylindrical grinding; coarse-pored wheels (structures 7 and 8)—for surface grinding of soft materials, and open wheels (structures 9 to 12)—for high-velocity grinding.

The standard system of marking grinding wheels adopted in the U.S.S.R. indicates the wheel manufacturer followed by the characteristics, size and shape. For example, the marking "Ilyich Plant 346CM25K=ПП500×150×305=35 m/sec" means that the wheel was made at the Ilyich Plant from aluminium oxide (artificial corundum) and has a grain size of 46, grade CM2, structure 5 and a vitrified bond. It is a straight wheel with an outside diameter of 500 mm, width 150 mm and bore 305 mm and the maximum permissible speed is 35 m per second. The structure is sometimes not indicated in the marking.

### 36-3. Grinding Machines

In accordance with their design and processing capacities grinding machines are classified as cylindrical grinding, internal grinding, surface grinding, single-purpose grinding, tool and cutter grinding and lapping machines.

The class of cylindrical grinding machines may be further divided into plain cylindrical (between centres), centreless, semiautomatic and automatic types; internal grinding machines are divided into chucking, planetary, centreless, semiautomatic and automatic types; surface grinding machines are divided into reciprocating and rotary table types, machines grinding with the periphery or the end face of the wheel, and semiautomatic and automatic types; single-purpose machines—into gear grinding, thread grinding, tracer-controlled grinding, spline grinding, ball grinding and other types; and tool and cutter grinders—into universal types for grinding various kinds of tools and special types for a particular tool. Besides these there are machine tools for processes that resemble grinding in

the use of an abrasive material for cutting. These include lapping, honing, superfinishing, polishing and buffing which use abrasive wheels, abrasive sticks and stones, coated abrasive products, polishing grains and pastes.

Each type of grinding machine is manufactured in several models.

The plain cylindrical grinding machine illustrated in Fig. 231 is designed for external grinding between centres. All the units of the machine are arranged on the base 1 which also houses the drive

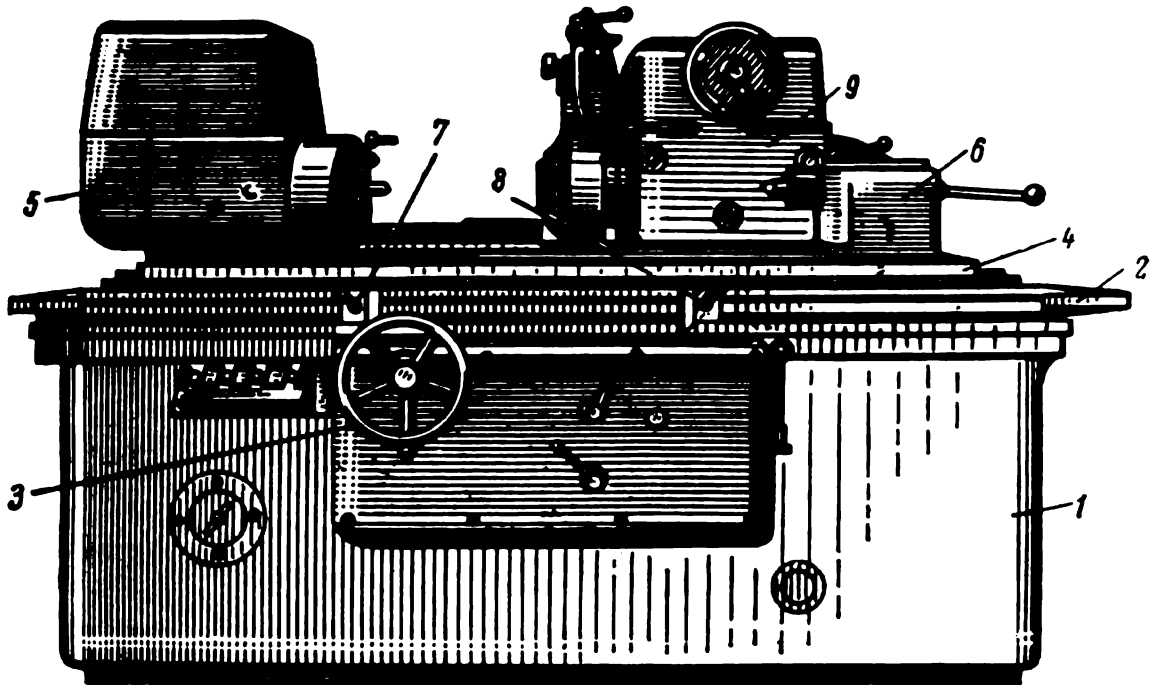


Fig. 231. Plain cylindrical grinding machine, model 3151

mechanisms and the hydraulic drive. Sliding table 2 is reciprocated by the hydraulic drive to obtain the longitudinal feed at a speed from 0.08 to 10 m per minute or it is traversed manually by hand-wheel 3 through a system of gearing. Swivel table 4 with T-slots for securing the headstock 5 and tailstock 6 is mounted on the sliding table.

The headstock serves to mount and drive the workpiece which is held either between centres or in a chuck. The work spindle in the headstock is powered by a motor located in the upper part of the headstock. In the model shown the work spindle has three speeds—75, 150 and 300 rpm.

The tailstock supports the rear end of work mounted between centres; its spindle can be advanced and retracted. The tailstock spindle with the centre is commonly advanced and clamped by a spring device. The tailstock can be adjusted along the slots of the swivel table to suit the length of the workpiece and can be clamped in the required position by bolts or an eccentric clamp.

Tapers are ground by setting the swivel table 4 to the taper angle in a horizontal plane in reference to the sliding table. Settings up to  $\pm 10^\circ$  can be made. Steep tapers are ground by swivelling the wheel head. The workpiece is mounted between the headstock and tail-stock centres and is rotated at a peripheral speed of from 20 to 30 m per minute.

Adjustable dogs 7 and 8 clamped in longitudinal slots provided at the side of the sliding table are set up to reverse the table at the ends of the stroke.

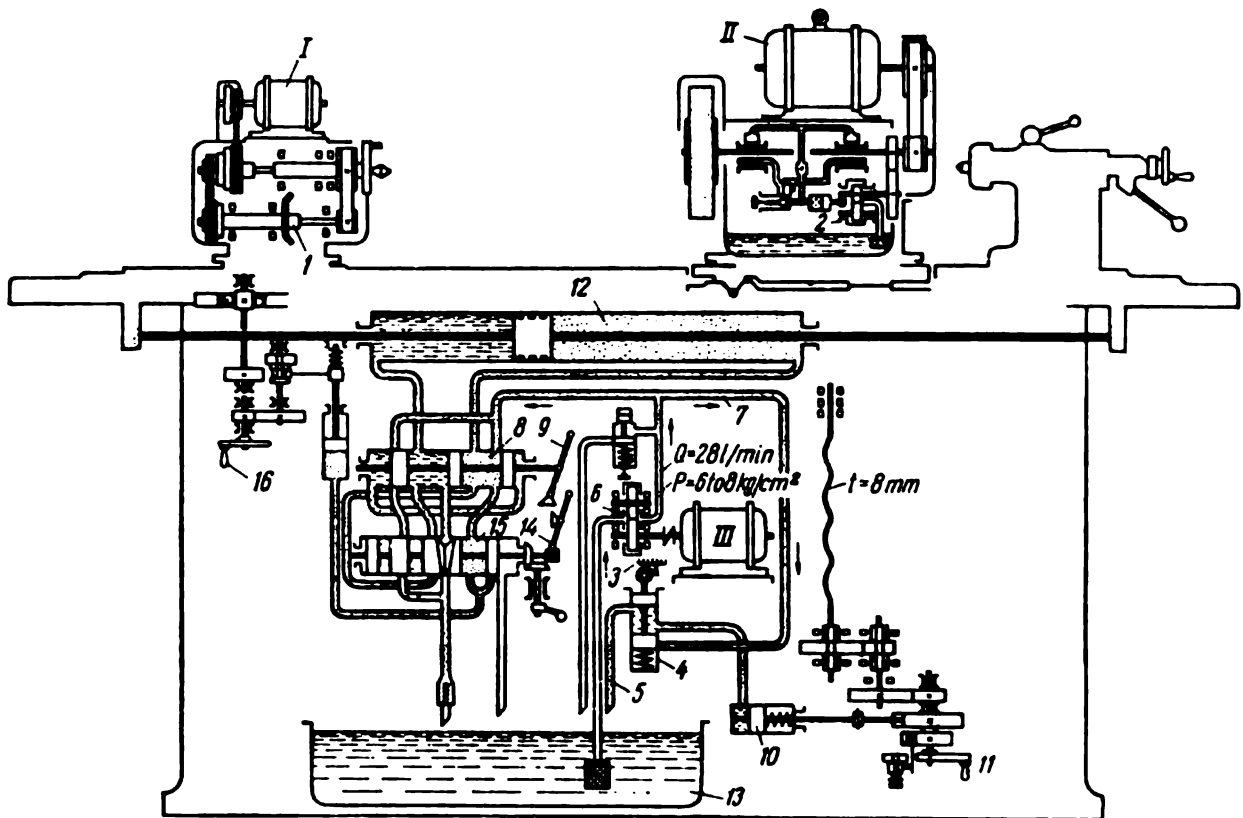


Fig. 232. Combined gearing and hydraulic system diagram of the model 315 cylindrical grinding machine:

1—headstock drive motor; 11—wheel head drive motor; 1—cone friction clutch; 2—spur-gear pump; 3—cam; 4—wheel head feed cylinder; 5—drain pipeline; 6—spur-gear pump; 7—main pipeline; 8—table travel pilot valve; 9—table travel reversing lever; 10—cross feed operating cylinder; 11—cross feed handwheel; 12—table travel cylinder; 13—oil tank; 14—hand control lever for table travel; 15—pilot valve for manual table control; 16—handwheel for manual table traverse

The grinding wheel is mounted and clamped on the spindle of the wheel head 9 which travels along cross ways at the rear of the base. The wheel is driven at speeds up to 1500 rpm from a motor in the upper part of the head.

Cross feed of the wheel after each stroke of the table is obtained by manually or hydraulically operated crosswise movement of the wheel head.

A combined gearing and hydraulic system diagram of the hydraulically operated cylindrical grinding machine, model 315, is given in Fig. 232.

The workpiece mounted between the headstock and tailstock centres is driven by motor *I* through a system of belts and the cone friction clutch *1*. The grinding wheel is powered from motor *II* through a single-stage belt drive.

Cross feed of the wheel at the end of each table stroke is operated by the reciprocation of the piston rod in cylinder *4*. The rod travels downwards due to the action of the bevelled cam *3* and forces oil into the operating cylinder *10* whose piston is pushed to the right and, by means of levers, rotates a system of gearing. Rotation of the gearing is transmitted to the cross feed screw with a pitch of  $t=8$  mm which is linked through a nut to the wheel head. Thus, intermittent cross feed of the wheel is accomplished.

Longitudinal feed of the workpiece is provided by the reciprocating movement of the table actuated either by the hydraulic drive or manually from a handwheel.

Adjustable dogs mounted on the sliding table trip lever *9* of pilot valve *8* during table reciprocation, turning the lever alternately to the right and to the left. As a result, the spool of pilot valve admits oil alternately into the ends of the table cylinder *12*. Under the pressure of the oil the piston and, with it, the table travel to the right and left alternately. This reciprocation of the table accomplishes longitudinal feed of the work. Lever *14* is provided for manual control of table travel; it operates the spools of pilot valve *15* which change the direction of oil flow and start or stop table travel.

Fig. 233 illustrates a horizontal-spindle reciprocating-table surface grinding machine. The base *1* contains the drive mechanisms and the hydraulic drive. Table *2*, actuated by rod *3* of the hydraulic table cylinder, reciprocates along ways on the base to provide the longitudinal feed. T-slots are provided in the table surface for clamping workpieces directly on the table or for clamping grinding fixtures or a magnetic chuck. The length of table travel is set up by adjustable dogs *4* which can be moved along and clamped in slots at the side of the table. By means of lever *5* the dogs reverse table travel at the ends of the stroke. Push-button controls *6* start and stop the machine.

Column *8* is secured to the base; it has ways for the vertical slide *9* which can be raised or lowered with the grinding wheel only manually from handwheel *11*. The vertical slide has horizontal ways for the wheel slide which houses the grinding wheel spindle. Horizontal crosswise movement of the wheel slide with the wheel, actuated either by hand from handwheel *10* or by the hydraulic drive, accomplishes the cross feed of the wheel. The components of the hydraulic drive of the wheel slide are housed in a recess of the vertical slide. Grinding wheel *7* rotates at constant speed; it is powered by a special built-in motor.



Surface grinding machines may be used for finishing long narrow surfaces as well as small workpieces set up, several at a time, on a magnetic chuck.

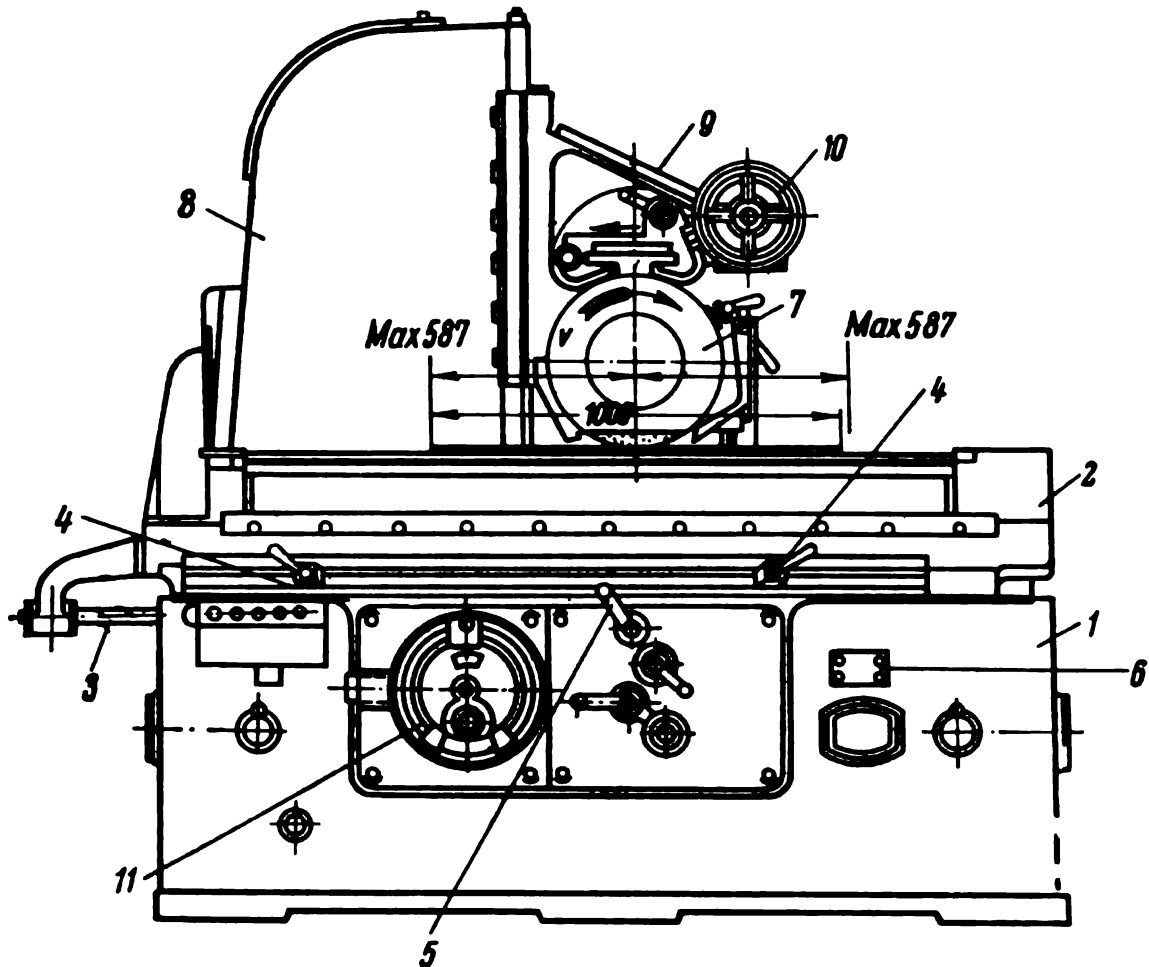


Fig. 233. Horizontal-spindle reciprocating-table surface grinding machine

Heavy work is ground on plano- or way-type surface grinding machines which resemble planers.

### 36-4. Principal Types of Grinding Operations

Operations which may be done on grinding machines include the grinding of external and internal cylindrical, tapered and formed surfaces; gear teeth; thread, and others, using appropriate wheels and fixtures for each job.

*External cylindrical surfaces* are ground on cylindrical grinding machines by the traverse, plunge-cut or full-depth method. Such surfaces can also be ground on centreless grinding machines by either the through-feed or infeed methods.

An example of through-feed grinding on a centreless grinding machine is shown in Fig. 234a. The cylindrical workpiece 1 is advanced along the loading rods 2 and 3 to the rotating grinding wheel 4

and regulating wheel 5 where it is drawn by the wheels into the grinding zone. Here the workpiece is supported by the work rest blade 6. After being fed through between the wheels by the action of the regulating wheel and ground, the workpiece runs out on guides 7.

The layer of metal removed by the grinding wheel in one pass reduces the diameter of the workpiece by from 0.02 to 0.3 mm.

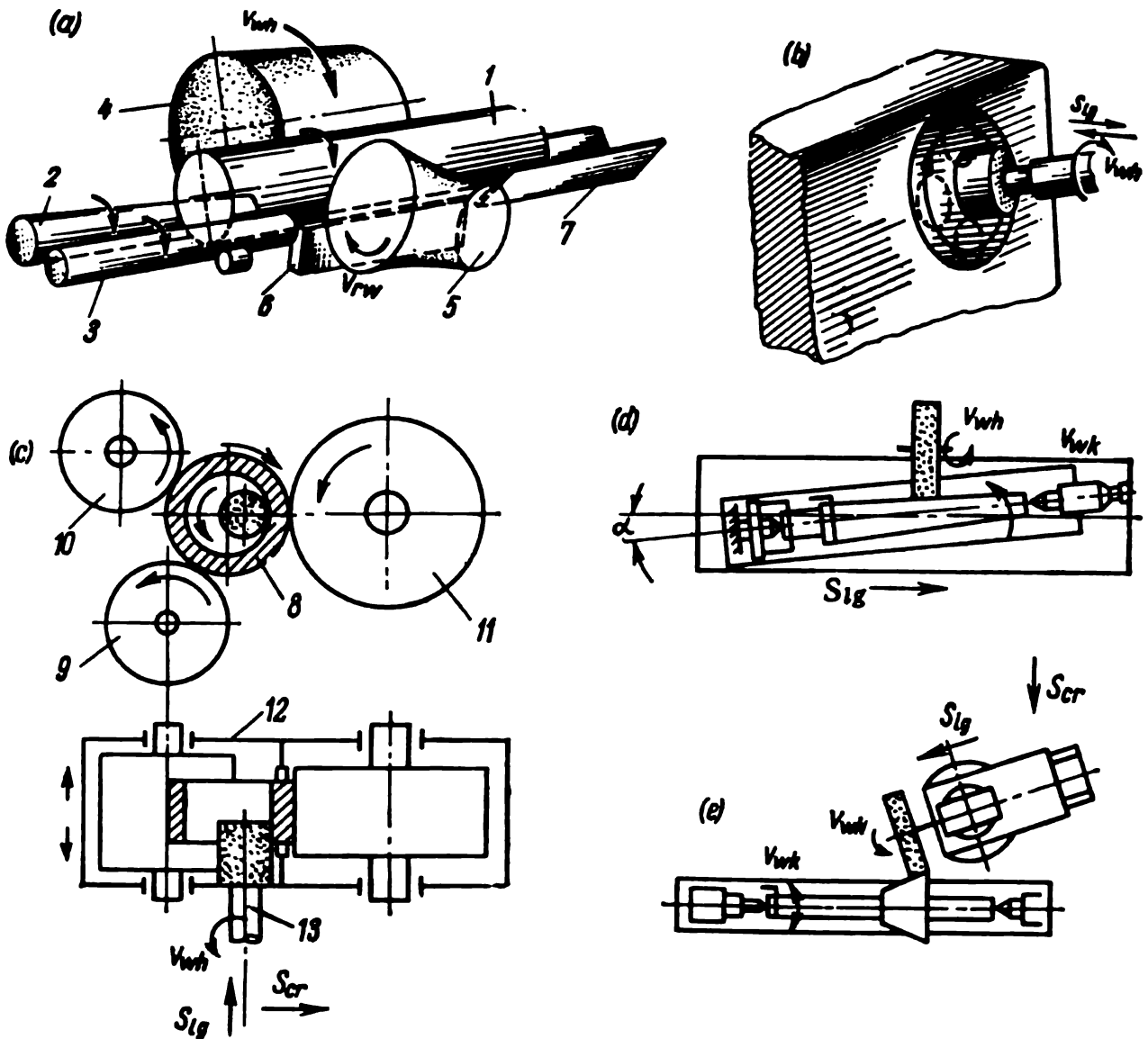


Fig. 234. Grinding external and internal cylindrical surfaces and tapered surfaces

Infeed centreless grinding is employed for stepped or headed work.

*Internal cylindrical surfaces* are ground on internal, planetary or internal centreless grinding machines.

A cylindrical hole in comparatively small parts can be ground by clamping the work in the chuck of an internal grinding machine where it is rotated by the work spindle (see Fig. 229b). A straight-type grinding wheel is rotated and has two feeds—longitudinal feed along the wheel axis and intermittent cross (radial) feed at the end of each pass.

The depth of cut depends upon the diameter of the hole being ground and may vary from 0.5 to 0.02 mm in roughing and from 0.002 to 0.01 mm in finishing operations.

An overtravel equal to  $\frac{1}{8}$  of the wheel width is required at each end of the stroke in internal grinding.

Heavy workpieces that are inconvenient or even impossible to clamp in a chuck are mounted on the table of a planetary grinding machine. In addition to rotation about its axis the wheel spindle

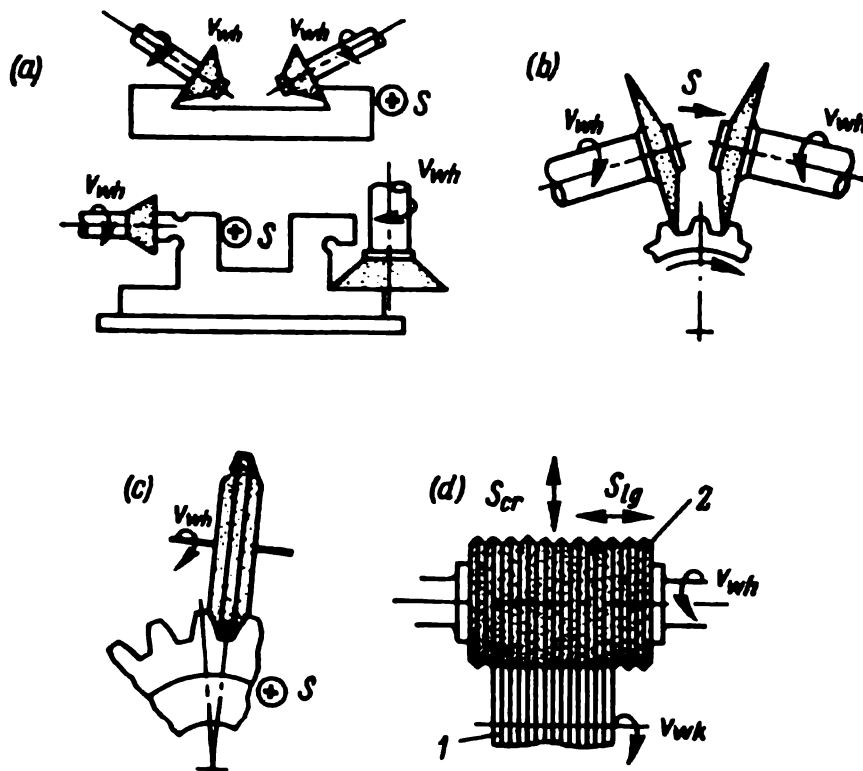


Fig. 235. Grinding complex ways, gear teeth and thread

of this type of machine also rotates with a planetary motion about the axis of the hole being ground (Fig. 234b). Axial motion of the wheel provides the longitudinal feed.

Holes in ring-type parts can be ground on internal centreless grinding machines (Fig. 234c). The workpiece 8 is loaded into the grinding zone where it is held between three steel rolls—support 9, pressure 10 and regulating 11—mounted in housing 12. The lever-mounted pressure roll 10 exerts a preset force on the workpiece holding it against the support and regulating rolls. The regulating roll, linked to the work drive, rotates the workpiece at a peripheral speed of from 40 to 60 m per minute. An axial force acting on the workpiece, due to the fact that the regulating roll axis is inclined to about one half of a degree, holds the workpiece axially with one end face against a roller stop. The hole is ground with a wheel mounted on spindle 13. The rotation of the work in grinding is the result of the friction force between the work and the regulating roll.

Very high accuracy can be maintained in centreless grinding.

*Tapered surfaces* are ground on cylindrical grinding machines by swivelling: (1) the table to the taper angle of the workpiece (Fig. 234d); (2) the wheel head to the same angle (Fig. 234e) or (3) the head-stock with the workpiece clamped in a chuck. In the second method longitudinal feed is obtained by the axial motion of the wheel.

In addition to the foregoing, internal tapered surfaces are ground on centreless machines by swivelling housing 12 (Fig. 234c) and the workpiece to the taper angle. Still another method of grinding external tapered surfaces is to true the face of the wheel to the required taper.

*Formed surfaces* are finished on cylindrical or surface grinding machines depending upon the type of workpiece to be machined. This class of job also includes the grinding of bed ways of complex cross section (Fig. 235a).

*Flat surfaces* are ground on surface grinding machines (see Fig. 233). Heavy workpieces are clamped on the table by means of pads, strap clamps and other devices or they are held in fixtures. Small work is usually held by a magnetic chuck.

*The teeth of gears* are ground on gear grinding machines either by the generating process or by a forming process in which formed wheels are used.

The generation grinding method makes use of two saucer-shaped grinding wheels (Fig. 235b) set so that their active surfaces (on the spindle side) are in planes tangent to the involute curvatures of two teeth on the gear. This is accomplished by truing the wheels to an angle equal to the pressure angle of the gear being ground. In grinding, the gear has a complex rolling (generating) motion.

In formed-wheel grinding of gears (Fig. 235c) the contour of the wheel is trued by a special fixture so that it coincides with the profile of the tooth spaces on the gear. The adjacent flanks of two teeth are ground simultaneously.

*Thread* is ground on thread grinding machines with either single-or multiple-rib wheels. Both internal and external threads are ground. In the grinding of external thread with a multiple-rib wheel (Fig. 235d) the work 1 is mounted between centres and is rotated at a definite speed. The grinding wheel 2 is mounted on the wheel head spindle and is rotated by a separate drive. The wheel also has a longitudinal (axial) movement which amounts to one pitch of the thread per revolution of the work.

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